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Canada Centre for Mineral and Energy Technology

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**Mining Research Laboratories**

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# ASSESSMENT OF AN INTEGRATED BLAST MONITORING PACKAGE FOR USE IN THE INVESTIGATION OF OPEN PIT PRODUCTION BLASTING PRACTICES

CANMET SCIENTIFIC AUTHORITY: YVES LIZOTTE

~~MRL 90-092~~ June, 1990

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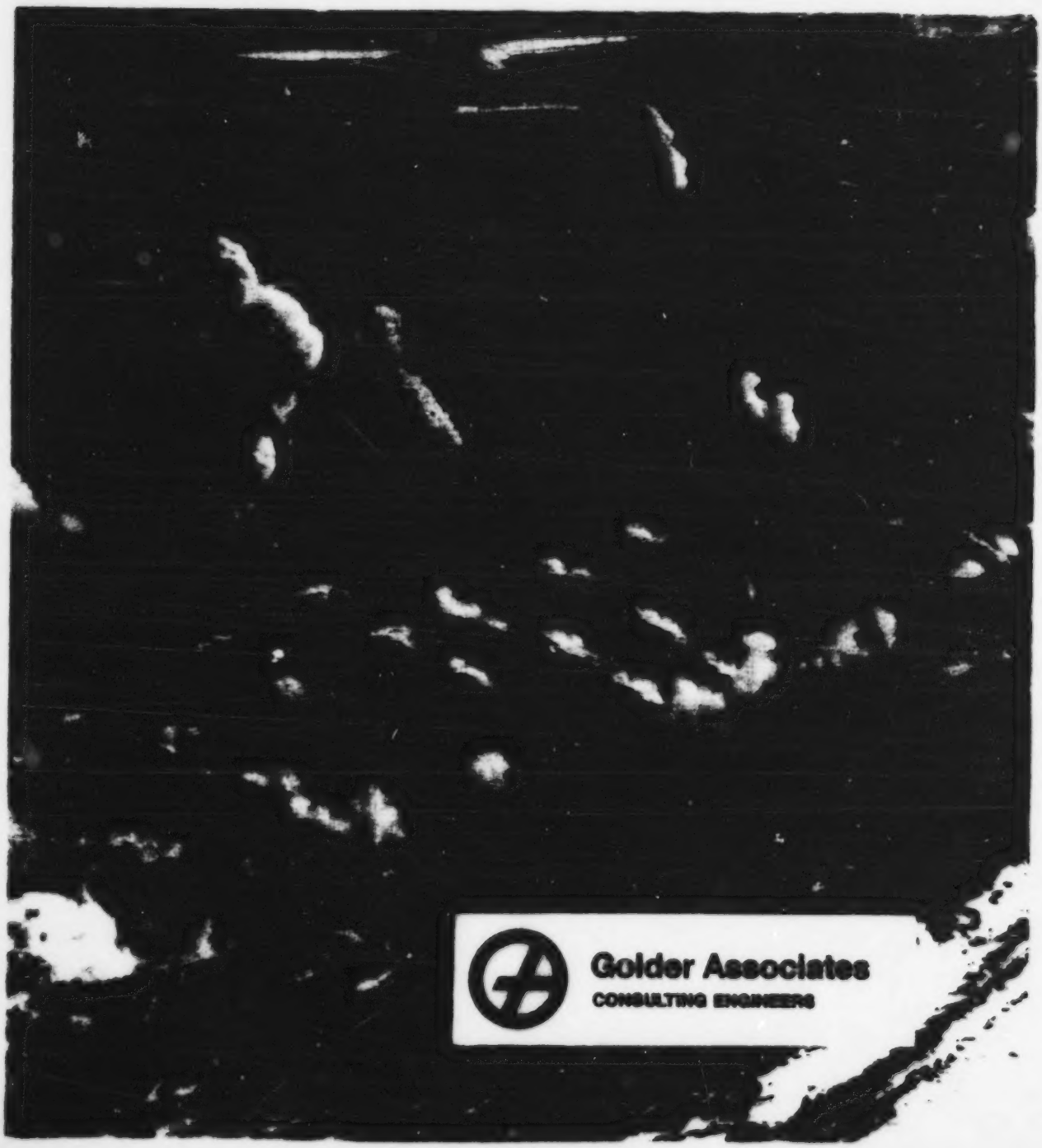
# ASSESSMENT OF AN INTEGRATED BLAST MONITORING PACKAGE FOR USE IN THE INVESTIGATION OF OPEN PIT PRODUCTION BLASTING PRACTICES

## FINAL REPORT



**Golder Associates Ltd.**
CONSULTING ENGINEERS

# ASSESSMENT OF AN INTEGRATED BLAST MONITORING PACKAGE FOR USE IN THE INVESTIGATION OF OPEN PIT PRODUCTION BLASTING PRACTICES

**FINAL REPORT BY:**

**GOLDER ASSOCIATES LTD.**
**VANCOUVER, B.C.**

Work on this project was conducted under the auspices of CANMET, Department of Energy, Mines and Resources.

DSS File Number: 09SQ-23440-9-9162

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CANMET Scientific Authority: Mr. Yves Lizotte

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July 11, 1990

CANMET
Energy, Mines and Resources, Canada
555 Booth Street
Ottawa, Ontario
K1A 0G1

ATTENTION: Mr. Y.C. Lizotte, Scientific Authority

**Re: Assessment of an Integrated Blast Monitoring for Use in the Optimization of Open Pit Production Blasting Practices**
**DSS Contract Number: 23440-9-9162/01-SQ**

Dear Sir:

Attached please find a copy of our final report on the completion of a joint industry/government sponsored study into the assessment of an integrated advanced blast monitoring package for the investigation of open pit blasting practices.

Two of the main objectives of the study; explosive velocity of detonation measurement and the use of conventional photographic techniques were successfully assessed. However, the assessment of high speed digital recording (BMX) was not fully completed. The nature of the rock mass and the blasting practices at this site inhibited full evaluation of this system.

It is clear that the development of complementary monitoring techniques could have a significant impact upon blasting economics through better understanding, and then optimization blasting practices. This is emphasized by the decision of Highland Valley Copper to continue the project with particular emphasis on reducing damage to the pit walls.

If you have any questions regarding this study, please do not hesitate to contact us.

Yours very truly,

GOLDER ASSOCIATES LTD.

D.P. Roberts

W.W. Forsyth

P.F. Stacey, P. Eng.
Principal

## TABLE OF CONTENTS


| | | Page |
|---|---|---|
| EXECUTIVE SUMMARY | | i |
| 1.0 | INTRODUCTION | 1 |
| 2.0 | BLAST OPTIMIZATION PROGRAMS | 2 |
| | 2.1 Current Methods of Blast Evaluation | 3 |
| | 2.1.1 Empirical Evaluation of Design Parameters | 3 |
| | 2.1.2 Conventional Vibration Monitoring | 5 |
| | 2.1.2.1 Analysis of Monitoring Results | 6 |
| | 2.1.3 High Speed Photography | 7 |
| 3.0 | ADVANCED BLAST MONITORING TECHNOLOGY | 9 |
| | 3.1 BMX System | 9 |
| | 3.2 Modern Photography Methods | 11 |
| 4.0 | HIGHLAND VALLEY COPPER | 12 |
| | 4.1 General | 12 |
| | 4.2 Geology | 13 |
| | 4.2.1 Engineering Geology | 14 |
| | 4.3 Mining System | 15 |
| | 4.4 Drilling and Blasting | 15 |
| | 4.4.1 Drilling | 16 |
| | 4.4.2 Explosives and Loading Procedure | 16 |
| | 4.4.3 Initiation System and Sequencing | 16 |
| | 4.5 Blasting Domains | 17 |
| 5.0 | FIELD PROGRAM | 18 |
| | 5.1 Phase I Field Program | 18 |
| | 5.2 Phase II Field Program | 19 |
| 6.0 | MONITORING RESULTS | 20 |
| | 6.1 Vibration Monitoring | 20 |
| | 6.2 Velocity of Detonation Measurements | 21 |
| | 6.3 Discussion | 22 |
| | 6.3.1 Digital Recording | 22 |
| | 6.3.2 Velocity of Detonation Measurement | 22 |
| | 6.3.3 Videotaping and Photography | 23 |

7.0 CONCLUSIONS 25
7.1 BMX Blast Monitor 25
7.2 Videotaping and Photography 26

8.0 OVERALL BLAST OPTIMIZATION PROGRAM 26
8.1 BMX Blast Monitor 26
8.2 Videotaping and Photography 27

REFERENCES 28

## LIST OF FIGURES

Figure 1 Blast Design Variables
Figure 2 Typical Peak Particle Velocity vs. Scaled Distance
Figure 3 BMX Sample Vibration Waveform
Figure 4 Plan of the Geology of the Valley Pit
Figure 5 Plan of Structural Domains of the Ultimate Valley Pit
Figure 6 Approximate Blast Locations (Valley Pit Plan)
Figure 7 Typical Blast Sequencing
Figure 8 Typical Blast Sequencing
Figure 9 Typical Monitoring Configuration
Figure 10 Typical BMX Waveform from this Site
Figure 11 Typical V.O.D. Measurement
Figure 12 Blast Induced Damage (Valley Pit Plan)

## LIST OF APPENDICES

Appendix I Blast Monitor Specifications
Appendix II Visual Recording
Appendix III Photographs
Appendix IV Blasting Practice Questionnaire

## 1.0 INTRODUCTION

This report presents the results of a joint Federal government - industry research project into open pit production blasting conducted at the Highland Valley Copper mine site, located approximately 50 kilometres southwest of Kamloops, British Columbia. The purpose of the project was to evaluate the applicability of an integrated blast monitoring system that emphasizes the use of a high speed digital recorder in the assessment and optimization of open pit production blast practices.

This project, which resulted from an unsolicited proposal dated July, 1989, was authorized by Mr. R. McMillan, Science Contracting Officer, Science and Professional Services Directorate, Department of Supply and Services (DSS) and Mr. Y. Lizotte, Scientific Authority, Department of Energy, Mines and Resources, CANMET-MRL Division under Supply and Services contract number 23440-9-9162/01-SQ.

The scope of the study was to evaluate the potential application of an advanced blast monitoring device known as the BMX Blast Monitor to open pit production blast optimization. The BMX unit was to be used in combination with a video camcorder and 35 mm still photography as part of an integrated system. It was not the purpose of the study to actually perform blast optimization at the mine.

The BMX Blast Monitor was developed by Blastronics Pty., Queensland, Australia. It is a digital system that provides monitoring records of the full vibration wave-train of geophones in the vicinity of a blast. The equipment has been proved to be invaluable in the optimization of blasting in several underground mining operations in North America, Australia and other countries. The application of the BMX to the larger production blasts that are used in major open pit mines had not been fully evaluated in Canada.

The study was conducted during two site visits in early 1990. Golder staff experienced in the use of the BMX equipment in underground applications monitored several production blasts during those visits. They were assisted by mine staff, notably Mr. D. Assmus, Senior Explosives Foreman and Mr. C. Yip, Planning Engineer.

The assistance of all Highland Valley Copper staff throughout the project is gratefully acknowledge. Without their advice and patience, the study would not have been possible.

## 2.0 BLAST OPTIMIZATION PROGRAMS

The objective of any mining operation is to produce the required tonnage of ore at minimum expense. In most open pit mines in hard rock, this involves a system of fragmentation of the rock followed by loading and hauling to the ore crusher or waste dump. In this system, the drilling and blasting required to produce the fragmentation necessary for efficient loading represents a significant proportion of the mining costs. There is therefore a strong incentive to optimize the blasting in terms of costs, and also in terms of minimizing damage that could result in instability in the pit walls.

Due to the potential gains from improvement in blasting practices, understanding the blasting process and optimizing blast design has become very important to any mining operation. Performance assessments and optimization of blasting in open pit mines are based upon the evaluation of the effect of specific design parameters on blast performance, in particular fragmentation, diggability and bench stability.

For a given rock type, design parameters which may influence blast performance include:

- explosive type
- powder factor and charge distribution (amount of explosive per unit of rock);
- blasthole size and layout (burden and spacing);
- initiation timing and sequence;
- direction of blasting relative to rock structure;
- freedom of the rock to move when blasted.

Poor compatibility of any or all of these parameters reduces the efficiency of the blast, and can even result in unsafe conditions. Typical problems can include: -

- misfires (part of the blast does not initiate);
- poor fragmentation;
- flyrock;
- excessive noise;
- damage to pit walls.

Careful assessment of the various parameters through blast evaluation programs can rectify these problems, and minimize blasting costs through improved fragmentation efficiency.

## 2.1 Current Methods of Blast Evaluation

### 2.1.1 Empirical Evaluation of Design Parameters

The typical means of evaluating blast performance include an empirical evaluation of the blast layout parameters. This can be augmented by vibration monitoring, and high speed photography. Combined, these provide a semi-quantitative evaluation process at best, and a system that is based on the implicit assumption that the explosives and initiation systems operate within the manufacturers' specifications.

The following discussion of the current blast evaluation methods serves also to describe the inter-relationship of some of the blast design parameters.

Relationships between the variables in open pit blast designs have evolved over a number of years, largely on the basis of field experience. These guidelines, or "rules of thumb", quantify the inter-relationship between design variables as presented in Figure 1.

Preliminary blast designs are typically established on the basis of the guidelines, and are then evaluated by field trials during the initial stages of a mining operation. During the field trials fragmentation, muck displacement and wall damage provide the main benchmarks by which the blasts are empirically evaluated.

Some of the main parameters for the blast designs are as follows:

Burden: the distance between the blasthole and the free face. The burden dimension is considered a function of charge diameter (which in turn is a function of hole diameter). The rule of thumb for burden calculation is:

$$B = \frac{25 \text{ to } 35 \times D_E}{12}$$

Where, B = burden (ft.) $D_E$ = explosive column diameter (in.)

Spacing: the distance between blastholes normal to the burden. The spacing is a function of the blasthole burden.

$$S = 1 \text{ to } 1.8 \times B$$

Where, S = spacing (ft.) and B = burden (ft.)

Subdrilling: the distance drilled below the bench grade to assure digging at bench grade. The rule of thumb for subdrilling is as follows:

$$J = 0.2 \text{ to } 0.5 \times B$$

Where, J = subdrill (ft.) and B = burden (ft.)

Stemming: an inert substance placed on top of the explosive to provide confinement of the explosive gases. An adequate stemming height is also essential to prevent flyrock. The required stemming is defined as follows:

$$T = 0.7 \text{ to } 1.3 \times B$$

Where, T = stemming (ft.) B = burden (ft.)

Explosive types - Explosive selection is controlled by the condition of the rock mass and the presence (or absence) of water. A poor quality rock will require an explosive with a high "heave" energy while a good quality rock will require an explosive with a high "shock" energy.

Free Face Geometry - The best fragmentation and face stability is achieved when the free face is parallel to the dominant fracture set. The newly formed face is then often a fracture face. Face stability is enhanced where the fracture sets are steeply dipping, as the volume of rock likely to fail along a bench decreases as the dip of the fracture sets increases.

Sequencing - The firing sequence of the blast should be such that an appropriate spacing to burden ratio is maintained; adequate time is allowed for burden movement and that individual blastholes are delayed sufficiently to avoid energy superposition.

### 2.1.2 Conventional Vibration Monitoring

The purpose of a conventional monitoring program is to establish a relationship between the maximum instantaneous charge (explosive weight per delay period) and peak particle velocity (PPV) at variable distances from a blast. Once a scaled relationship between these two parameters is established for a given rock type, blasts can be designed to minimize blast vibrations transmitted into the pit slopes. As such, the conventional vibration monitoring is largely a tool for reducing wall damage, rather than optimizing production blasts.

Typical vibration monitoring instruments in this category have the following characteristics:

| | |
|---|---|
| Frequency response: | 2 to 250 Hz +/- 3 dB |
| Recording rate: | 1 kHz |

| | |
|---|---|
| Recording times: | 1 to 7 seconds in 1 second intervals |
| Recording resolution: | 0.124 mm/sec. |
| Recording ranges: | up to 200 mm/sec. (velocity)<br>up to 30 g's (acceleration-calculated)<br>up to 38 mm (displacement-calculated) |
| Trigger source: | Threshold or manual |

(Information for Instantel DS-477)

The limitation of the instrument as far as blast analysis is concerned is that the sample rate (1 kHz) does not permit identification of individual events. The time separation between individual events and their high frequency (and hence short duration) at a 10 - 30 m recording distance do not allow an instrument of this type to be effective.

2.1.2.1 Analysis of Monitoring Results

The output from the vibration monitor will contain the three components (radial, transverse, vertical) of velocity and a vector sum of these components (PPV). The peak particle velocity (PPV) is then plotted on a log-log graph relating PPV and scaled distance as shown in Figure 2.

The scaled distance is calculated as:

$$\text{Scaled Distance} = D/\sqrt{w}$$

After a number of blasts have been monitored, and the results plotted, it is then possible to assess the plotted data. A linear regression analysis is used in order to obtain the variables k and $\beta$ in the relationship:

$$V = k\,(D/\sqrt{W})^{-\beta}$$

where: V = peak particle velocity
k = intercept at unity scaled distance
D = distance
W = explosive weight per delay
$\beta$ = slope

Once this relationship has been determined for a given rock type, a blast design policy can be established for controlled blasting designs.

For example, given a vibration limit of 250 mm/sec. and a distance to the blast of 30 m, the maximum allowable weight of explosive that can fired per delay period is 775 kg.

This method of blast design is limited by the following:

- the method assumes that delay separation is sufficient to eliminate inter-hole vibration enhancement;

- the method assumes that a blasthole acts as a point source rather than acknowledging the VOD of the explosive and hence is invalid in the near field.

2.1.3 High Speed Photography

The use of cameras capable of filming at speeds of up to 500 frames per second can provide detailed information about the complex interaction between the explosive and the rock mass. The application of high speed cameras is fairly wide spread and well documented. A good operational discussion is given by Chiappetta et al (1983) where it is stated that the information obtained from high speed photographic analysis can be categorized into two main groups. These are as follows:

- Qualitative

- First rock movement
- Firing sequence of holes in blast
- Confinement and/or blowout of stemming
- Shape or form of primary rock movement
- Confirming function of explosive accessories

- Quantitative

- Firing time of initiators
- Time and duration of escaping gases
- Acceleration, direction, and velocity of projectiles
- Initial time of detonation
- Ground swelling velocity
- Stemming ejection velocity
- Total flight time of a projectile
- Casting range of fragmented material
- Confinement time of burdens, bench top, and gases after hole detonation

When combined with general information on the blast layout, these data have proven useful in helping to determine the following:

- cause of misfires
- poor loading practices
- poor firing practices
- effects of redrills, water, gas venting, and buffers
- proper delay interval between holes and row of holes to obtain adequate forward burden relief
- optimum initiating system
- massive ground movement
- source of oversize
- optimum cast of fragmented material
- optimum explosive-rock-burden combination
- final muckpile geometry
- origin
- components
- operation
- advantages (theoretical as proven in underground work)

Although high speed photography does not give a complete analysis of a blast, it does have advantages over such traditional techniques as computer modelling and laboratory testing where compromises and assumptions must be made. Photography is particularly useful when used in conjunction with other blast evaluation methods as it provides a visual reference of sequence of events in the blast.

## 3.0 ADVANCED BLAST MONITORING TECHNOLOGY

The scope of this research program was to assess the integrated use of more conventional blast monitoring methods with an advanced blast monitoring system. This advanced system enables more precise evaluation of blast initiation and performance than traditionally available.

### 3.1 BMX System

The Blastronics BMX Blast Monitor was designed as a multi-purpose blast monitor, capable of providing both near field information concerning the performance of separately delayed charges, and far field information concerning blast induced vibrations (both air borne and ground borne) in buildings and structures remote from the site of blasting and in the ground itself. For each application, the monitor provides a full time history of the event, and this information can be simply interrogated and analyzed by the blast operator or mine engineer. The data recorded by the monitor may be produced by almost any type of gauge, including geophones (velocity gauges), accelerometers, strain gauges, microphones, Velocity of Detonation (VOD) gauges, extensometers and other displacement gauges.

To achieve such versatility in operation, the Blastronics BMX Blast Monitor is highly portable and highly interactive in operation, has large memory (up to 64 kilobytes), can sample at speeds of up to 200,000 samples per second, and has up to 10 recording channels. The input channels can be configured in any desired manner, so that any combination of gauges can be used, according to the particular application of the user.

The Blastronics BMX Blast Monitor is programmed within a few minutes, and is then generally located approximately 50 m away from the blast. Sensors may be positioned in many different locations around the blast or structure being monitored, including down specially drilled boreholes located 10 to 30 m behind the blast (near field); signals from each instrument are then routed to a common point at a junction box. All signals are then transferred to the monitor via a single trunkline cable. The digital

recorder is triggered either by a wirebreak (wire tied to the starting detonator or cord), or by a signal threshold selected by the operator from a menu. In this application, the wirebreak is the safest and most reliable means of recording, since only a break in the wire will trigger the recorder, and the monitor can safely remain in an armed and ready state for several hours prior to the blast being fired.

In the far field application, where the distance between the blast and the monitor point is too far to permit use of a wirebreak, the threshold triggering must be used. Any of the input signals can be used to trigger the monitor, and a range of threshold values can be selected by the operator.

Data are displayed on the computer screen within approximately 2 minutes of returning to the monitor after a blast. The full vibration waveform, (A), is displayed at the top of the computer screen, showing the main detail of the record as shown in Figure 3. Specific features of this full record are investigated in greater detail using the high resolution "zoom" function of the computer software. The horizontal bar cursor, (C), below the 64 k waveform represents the size and location of the lower waveform, (B), from within the full waveform. By varying the size and location of the bar cursor "window", views of varying detail can be obtained of any section of the full waveform.

Amplitude and timing data are obtained from the "window" waveform by use of the vertical cursor (D, Figure 3). This cursor can be moved in either direction along the waveform, and the co-ordinates of the waveform at the cursor location are displayed at the top of the computer screen, (E). Maximum and minimum levels are obtained quickly using a single keystroke search routine. By moving the cursor to each impulse in Figure 3, the precise firing times of all charges can be obtained, together with the maximum vibration amplitudes for each charge. The actual firing times can be compared with the nominal or designed firing times by superimposing the designed firing times on the computer screen.

Multiple waveforms can be displayed on the screen representing data from different sensors within the same blast, or representing data from completely different blasts. Each waveform can be analyzed totally independent of the others. If two or three orthogonal components of a triaxial gauge are displayed, the resultant vectors sum can be

obtained by a single keystroke using the optional math functions. The vector sum is also displayed as a full vibration waveform which can be interrogated using cursors or the optional special math functions in the same manner as any other waveform. Paper copies of waveforms are obtained from a graphics printer using a single keystroke.

Technical specifications of the instrument are presented in Appendix I.

Numerous studies related to underground blasting, and specific open pit applications that have been assessed in Australia, have shown that the system can take traditional blast monitoring methods to more advanced levels of understanding.

## 3.2 Modern Photography Methods

Visual recording can be used to provide a better understanding of blasting in two ways. Firstly, it can help to determine if the blast is working as designed; that is, that the sequence of initiation, ignition, detonation, etc., is the same in practice as on paper. Secondly, it can record the response of the rock mass to the explosive action so that the execution of the design can be analyzed. This would include stemming performance, ground movement, misfires, fragmentation, back-break etc.

High speed photography is both time consuming and expensive; an inexpensive means of visually recording and reproducing the above aspects of the blast was required. It was felt that recent advances in amateur photographic equipment had the potential to provide economical, easily obtainable records. The following two devices were evaluated to determine if sufficient resolution and detail could be collected:

- The camera used was a Canon EOS 630, a 35 mm format camera capable of producing 5 frames per second. When blast filming, this was used in conjunction with a 70 mm to 210 mm variable focal length lens which provided sufficient versatility to accommodate variables such as blast size, filming distance and pit geometry. Even under poor lighting conditions, a shutter speed of 1/1000 of a second could usually be achieved resulting in high quality prints.

- In addition to photography, a Canon H660 video recorder was used. This employs the hi-band 8 mm video system to produce 30 frames per second. A camcorder has a few distinct advantages over traditional VHS recorders, namely, higher resolution (525 vs 480 lines) and weight. Again, a shutter speed of 1/1000 of a second was found to be achievable even under poor lighting conditions. This resulted in high quality resolution for frame by frame scrutiny.

For visual recording, two parameters are of primary importance; location and timing. Ideally, filming should take place from a point well above the blast and opposite from the initial detonation point. This will provide an excellent record of hole ignitions and detonations which will not be immediately obscured by the vapour, gases, stemming, fly rock, etc., expelled by the first few holes. Experienced mine personnel were found to be indispensable for recommending ideal vantage points which were safe and accessible. In photography, timing was also critical as 36 frames at 5 frames per second allows 7.2 seconds of filming. Again, mine personnel were helpful by providing a second by second countdown previous to blasting.

For purposes of a cursory blast analysis, photographs were found to be excellent. The timing of events can be quickly estimated to the nearest 0.2 seconds (200 milliseconds) and general blast performance easily seen, shown and/or reproduced. Video was found to be useful for several reasons. Since no developing is required, it can be viewed within seconds of the blast. Also, as there are more frames per second than photography, it provides a more accurate record of the sequence of events. Video also gives an excellent indication of ground movement, particularly when viewed in "slow motion".

## 4.0 HIGHLAND VALLEY COPPER

### 4.1 General

Highland Valley Copper is a partnership between Cominco Ltd., Rio Algom Ltd., Teck Corp. and Highmont Operating Co. It is located approximately 50 kilometres south west of Kamloops, British Columbia and operates an open pit copper mine with molybdenum as a by-product. The mine operates at a rate of 268,500 tonnes per day of

ore and waste material from two pits, Valley and Lornex, and produces in excess of 350 million pounds of copper per year.

The Valley pit is pictured in Photograph 1. Current pit dimensions are approximately 1500 metres N-S, 1800 metres E-W and approximately 200 metres deep.

## 4.2 Geology

The Valley Pit is located on the west side of the northwest-southeast trending Highland Valley at an approximate valley floor elevation of 1200 metres above sea level.

The Valley orebody occurs in the porphyritic quartz monzonites and granodiorites of the Bethsaida phase of the Guichon Creek Batholith. The rocks within the Valley Pit are relatively competent, and show well defined joint and fault patterns.

The Valley orebody occurs at the intersection of two major regional faults, see Figure 4. The Lornex Fault is an approximately 200 metre wide, north-south trending fault zone that defines the eastern limit of the orebody and dips steeply to the east. The Highland Valley Fault strikes northwest-southeast, is truncated by the Lornex Fault and does not appear to extend into the orebody. The existence of a third major fault known as the Lake Fault has been postulated, this fault may intersect the northwest portion of the orebody.

Two major shear zones and a number of fracture zones have been defined within the limits of the ultimate Valley Pit. The Osprey Shear Zone is an approximately 200 metre wide, north-south trending shear zone on the east side of the pit, just west of the Lornex Fault Zone. This zone is also interpreted to dip to the east at approximately 60 degrees. The Southwest Shear Zone is a 260 metre wide, east-west trending shear zone that intersects the southwest corner of the pit and dips to the south at approximately 45 degrees. Three zones of north-south trending intense fracturing with steep easterly dips have been identified in the central and western portion of the pit.

4.2.1 Engineering Geology

Structural geology mapping of the Valley ore deposit was conducted during the exploration and development programs, and has been continued on a routine basis during mining. The results of structural mapping obtained from the following data sources have been used to determine the orientation of the major geologic structures within the Valley Pit:

1) Golder Associates Report V6912-A "Report to Cominco Ltd. on Pit Slope Design in Rock, Proposed Valley Copper Open Pit, Highland Valley, British Columbia."

2) Report on "Geology and Ore Deposits of the Highland Valley Camp", W.J. McMillan, Geological Association of Canada Field Guide and Reference Manual Series Number 1. (Contains orientation of major structures based on the results of approximately 14,000 structural measurements collected in the exploration adits by Cominco Ltd. geologists.)

3) Structural mapping data collected by K. Newman (HVC staff geologist) between 1982-1986 from the initial pit expansions.

4) Structural mapping data collected by J. Armstrong (summer student) during the summer of 1984.

5) Structural mapping of Pit 3,4 and 5 walls conducted between 1986 and 1989 by S. Daly (HVC staff geologist).

The structural mapping data indicates that in general there are five major structural domains within the Valley Pit. An analysis of the structural data shows a slight variation in the orientation of major structures between the northern (North Structural Domain) and southern (South Structural Domain) sections of the pit. Along with the two major domains are three further domains; East Structural Domain which is similar to the North Structural Domain but includes steeply dipping structures that parallel the Lornex

Fault, the Lornex Fault Zone and Osprey Shear Zone. The domain locations are shown in Figure 5.

The majority of this project (all but one blast) was conducted in the South Structural Domain. The orientations of the major structures in this domain have been identified as 80°/080°, 60°/080°, 85°/030°, 55°/030°, and 45°/135° (Dip/Dip Direction).

## 4.3 Mining System

The existing Valley Pit has been developed in a series of push-backs with the walls of the third, fourth and fifth push-backs currently exposed. The current pit floor is at elevation 1112.5 metres and the crest of the fifth push-back is at elevation 1337.5 metres. The ultimate design pit floor is at 712.5 metres and the ultimate pit crest is at 1450 metres. The inter-ramp slope angles vary between 38 and 45 degrees. The pit is excavated using a conventional truck and shovel mining operation. The rock is fragmented prior to excavation by the drilling and blasting of 12.5 metre high operating lifts. Half of the rock mined is ore grade material which is fed through an in-pit crushing system consisting of two semi-mobile 60 inch by 89 inch gyratory crushers prior to being transported to the mill area by conveyor. Therefore, a small run-of-mine fragment size is required to ensure high crusher throughput.

## 4.4 Drilling and Blasting

As the Valley Pit is being excavated as a series of concurrent expansion shells, each with a mining width of approximately 150 m, the current procedure is to blast the outside section of the bench in a series of "production" blasts that progress around the bench, leaving a 30 to 50 m wide strip along the wall which is mined later as a "buffer" shot. This concept is illustrated in Figure 6.

Where possible, the production blasts are fired using two free faces, i.e. the pit wall below and the adjacent bench face, e.g. Blast 52 in Figure 6. However, production constraints and equipment availability frequently prevent mucking of the previous blast before the next blast is fired, e.g. Blast 25. The result is a relatively "choked" condition. In extreme cases, e.g. Blast 32, no free faces are available beyond the bench surface.

An additional factor in the blast designs is a general requirement that fragmentation and muckpile looseness must be sufficient to provide for optimum shovel and crusher productivity. "Choked" blasting is considered by operating personnel to be one means of ensuring good fragmentation.

### 4.4.1 Drilling

Drilling is carried out using Bucyrus-Erie 45-R drills equipped with 14 metre (46 ft) drill strings and 270 mm (10.625 in) bits. All patterns are staggered with a burden to spacing ratio of 1.15 in order to obtain an even charge distribution throughout the rock mass. Exact burdens and spacings are determined by the "blastability" of the rock (see Section 4.5) with the survey department marking the position of each hole to be drilled. Blastholes are drilled to a nominal depth of 12 to 14 m depending upon the bench grade.

### 4.4.2 Explosives and Loading Procedure

All production holes are loaded with a site blended Ireco emulsion product. This product has an approximate density of 1.3 gm/cc, and a relative bulk strength of 129. Holes are loaded with between 475 and 575 kilograms of emulsion to leave a 6.4 metre collar. Although the preferred stemming material is crushed rock, blastholes are generally stemmed with drill cuttings due to the costs involved with creating and transporting the crushed material.

### 4.4.3 Initiation System and Sequencing

Blasts are primed with a combination of downhole Nonel XT Millisecond delay detonators and surface delays. Primers are suspended approximately 2 metres from the hole bottom (roughly the new bench elevation). Downhole delay elements between #5 and #18 are used together with surface delays of 35, 50 and 85 milliseconds. Blast sequencing is determined by either the blasting foreman or the lead hand on the blasting crew. In general, blasts are initiated from one corner or from the center in a V-sequence, depending on the geometry of the blast and the number of free faces available. This practice modifies the drilled spacing to burden ratio of 1.15 to an effective spacing to

burden ratio that varies between 2 and 3. Examples of blast sequencing are presented in Figures 7 and 8.

## 4.5 Blasting Domains

Over the past years of operating, Highland Valley Copper has developed a system of dividing the pit into different zones of blastability. This system is based upon the following criteria:

- argillic alteration intensity
- fracturing/faulting intensity and alteration
- silicification
- response to hammer blows (field strength test)
- fragmentation size distribution
- high toe areas that correlate with high strength zones
- diggability
- drill penetration rates (blasthole, drainhole and diamond drill)

Both blasthole patterns and powder factors are changed according to the relative blastability of the specific domain. The drilled burden to spacing ratio remains fixed at 1.14, the pattern is merely expanded or contracted for each blasting domain. Typical burdens and spacings for the different domains are given in the table below:

| Blastability | Burden (m) | Spacing (m) | Powder factor (kgs/tonne) |
|---|---|---|---|
| Good | 9.1 | 10.4 | 0.182 |
| Moderate | 8.7 | 10.0 | 0.197 |
| Poor-1 | 8.5 | 9.7 | 0.209 |
| Poor-2 | 8.0 | 9.2 | 0.235 |
| Poor-Blocky | 7.8 | 9.0 | 0.245 |

Blasting domains are determined by the site Geotechnical Engineer, Mr. Sean Daly. From his data a number of maps are produced, ranging from an overall

pit representation to individual bench plans. Blasthole layouts are determined using these plans. The number of available free faces is not taken into account when designing blasthole patterns.

## 5.0 FIELD PROGRAM

The field program was divided into two phases, with the first phase from February 20 to March 1, 1990 and the second phase from March 12 to 20, 1990. This planned division of the field program was to take into account the many unknowns that existed prior to beginning the research. If any unforeseen problems had arisen, a period of time was allotted in order to deal with them. This approach proved its worth when problems developed with the velocity of detonation (VOD) measurement equipment, necessitating a trip back to Vancouver.

The field programs involved observing blasting practices, reviewing geologic data, installing instrumentation, monitoring and photographing blasts, and measuring VOD.

The program was conducted around routine production blasting at the mine. However, as with any operation, the scheduling of any mining operation was subject to changes due to equipment breakdown and production constraints. For this reason the number of monitored blasts, unfortunately, was lower than expected, and the demands of maintaining production also meant that smaller scale blasts were not possible. Further, the single hole blasting that was proposed, and which would have played an important role in the optimization process was also precluded.

### 5.1 Phase I Field Program

The first phase included monitoring two production blasts, making two VOD measurements, and video taping and photographing five blasts. Individual summaries for each blast are given in Appendix II. The blast monitoring locations are outlined on Figure 6.

The **blast monitoring** utilized fully grouted geophone arrays placed at 18 m behind the last row of blastholes (roughly two burdens). This generally places the instruments outside the zone of influence of the blast and allows accurate measurements to be made. The accurate interpretation of the recorded signal is dependent upon a near elastic response of the rock mass to the incident energy, therefore instruments must be placed as close as possible to the blast but out of the damage zone. In this case however, the damage zone extended up to 30 m behind the last row of blastholes. The result of this damage was destruction of the instrument arrays. This monitoring distance was tried twice to eliminate the possibility of a geologic anomaly.

**Velocity of detonation** (VOD) measurements were carried out using an in-hole sensing ribbon. This ribbon is attached to the booster and makes measurements from this point upward through the explosive column. The VOD measurements were initially made in the last row of blastholes. The same problem that hindered vibration monitoring also hampered VOD measurement. The VOD probes were being sheared prior to recording; therefore, no successful measurements were made. A junction box must be placed 20 m from the VOD probes; this precluded probe placement in the initial firing blastholes.

The videotaping and photographing was completed with no major problems. This was due to the help of Willis McBride, Training Officer, who located all photography/video taping sites.

A typical blast and the applied instrumentation is shown in Figure 9, and a typical sequence of photographs is presented in Photograph 2.

## 5.2 Phase II Field Program

The second phase of the field program involved monitoring one blast, making five VOD measurements, videotaping and photographing six blasts. The monitoring locations are shown in Figure 6.

The results of the Phase I indicated the need to make changes to instrument placement in both the vibration monitoring and VOD measurement.

The **blast monitoring** instrument arrays were moved from 18 to 30 m from the last row of blastholes. This move allowed complete signal recording and eliminated the destruction of the instrument arrays. Although the blast induced damage did not destroy the instruments, there was significant damage up to and beyond the arrays and hence the vibration signal travelled through significantly disturbed rock. This damage and the large source to sensor distance precluded detailed interpretation of the recorded signal.

The **VOD** probes were placed in corner holes of the front row of blastholes. The initiation sequence was slightly modified to ensure that adjacent blastholes did not initiate prior to VOD measurement. This strategy worked well and four successful measurements were made.

Videotaping and photographing were carried out with no problems.

## 6.0 MONITORING RESULTS

### 6.1 Vibration Monitoring

The results of the vibration monitoring using the BMX equipment as described in Section 3.1 and Appendix I can be summarized as follows:

The first two blasts (Phase I) were monitored with two biaxial geophone arrays fully grouted at a distance of 18 metres from the last row of blastholes (roughly two burdens). In an underground mining environment this separation is generally sufficient to prevent high vibration levels and ground displacement from damaging the instrument arrays. However, in these two cases sufficient ground movement took place (up to 1 m of displacement at this distance) to disrupt signal transmission. This distance was tested twice to account for the possibility of a local geologic structure. The third blast (Phase II) was monitored at a distance of 30 metres.

At a distance of 30 m, it was possible to record for the full blast duration without destruction of the sensors. This increased distance between the blastholes and the instruments did, however, have a negative impact upon the monitoring results. Although not totally responsible for the poor signal quality shown in Figure 10, the greater travel

distance through the disturbed rock plays a major role in decreasing signal quality. The energy from each blasthole must pass through more of the rock mass, much of which has been previously damaged, where it can be attenuated at discontinuity boundaries and consumed during displacement along these discontinuities. The energy "signature" is therefore markedly distorted when it reaches the sensors.

A problem common to both monitoring distances was the large number of holes that were fired in any given blast, up to 220. When this number of holes are detonated in a relatively short time span (approximately one second), it becomes very difficult to separate individual events. It should be noted that there is also likely to be considerable enhancement of vibration levels between holes firing so close together. Coupling this problem with the high degree of signal attenuation, resulted in signals that provided vibration levels for the whole blast rather than individual blastholes.

The recorded vibration levels are presented in Table 1. A comparison is made to typical data collected at other mines to examine the potential for wall damage due to blasting operations. This comparison is presented in Figure 2.

### 6.2 Velocity of Detonation Measurements

An assessment of the explosive product was carried out using in-hole VOD measurement probes. The following are results of those measurements:

Initially the VOD measurements were tried in holes at the back of the blast pattern. The large displacement of ground due to earlier firing blastholes severed the VOD probes prior to measurement and no signal was received. Further measurements were made in front holes, and the delay sequence was modified such that the holes to be measured were protected from adjacent holes prior to detonation.

The results of the VOD measurement are presented in Table 2, and illustrate the potential variability of the detonation of the emulsion up the hole. A typical blasthole and the associated VOD measurement is shown in Figure 11. The degradation of the velocity of detonation near the top of the hole may be due to contamination of the

explosive by the stemming material or alternatively by fuel/oxidant separation in the explosive. However, more holes would have to be tested to confirm this possibility.

## 6.3 Discussion

### 6.3.1 Digital Recording

The high speed digital recorder is a superior instrument when compared to traditional vibration monitors. However, despite great gains in the ability to interpret complex blast geometry, the instrumentation could not separate the up to 200 individual events at nominal 5 to 10 millisecond intervals given a monitoring distance of 30 metres and a blast damaged rock mass. This does not imply that the recorder is not of use in the optimization process, only that the blasts must be scaled down and adjacent damage reduced if the data recorded by the BMX unit is to be decipherable.

The ability to interpret the vibration waveforms is controlled by the source to sensor distance, the initial time separation of energy sources and the nature of the rock mass (including the damage that it undergoes). The accuracy to which energy levels and frequencies can be quantified attributed to individual sources is dependent upon near elastic response of the rock mass to the incident energy. Considerable attenuation of signal transmission can occur in a competent rock mass over a large distance, or in a poor quality rock mass over a short distance. The quality of the signals recorded during this program would indicate that the relatively competent rock mass is being degraded by blast energy. In general, the signals were consistent with those recorded at other sites in poor quality rock.

The vibrations that were recorded are high (up to 450 mm/s) given the source to sensor distance (18-30 m). This energy was probably responsible for the large scale damage that was observed (see Photographs 5, 6 and 7).

### 6.3.2 Velocity of Detonation Measurement

The velocity of detonation (VOD) measurement instrument worked well when the operating conditions allowed signal recording. The VOD measurements provided one of the most important characteristics of the explosive product.

The VOD for the explosive product was found to be variable, see Figure 11 and Table 2. This could be due to slightly different batch mixes, or possibly stemming contamination. The probability that the appreciable drop in VOD for the upper approximately 1.5 m of the column was due to stemming contamination is high given the density difference (2.6 vs. 1.3 gm/cc) and the drop height (6 metres). A further but less likely explanation could have been dilation of the fractures in the upper section of the hole, reflecting the use of front row holes for the monitoring.

6.3.3 Videotaping and Photography

The visual recording of the blasts provided a considerable amount of information, including: -

- Initiation of surface lines (Photograph 2);
- Initiation of individual holes and general blast development (Photograph 2);
- Ejection of stemming material (Photograph 2);
- Energy transmission through the rock (video only);
- Crack development behind the blast relative to blast development (Photograph 2);
- Nature of the muckpile (Photograph 3).

The camcorder and 35 mm SLR camera provided similar information, the difference primarily being the continuous nature of the camcorder and the hard copy provided by the SLR. The primary difference between the two "amateur" techniques and high speed photography is the frequency of sampling. High speed photography would have allowed more detailed assessment face movement, but in this case, face movement was not considered overly important since the muck does not move a great distance at this mine.

The production blasts at Valley are designed to ensure that muck size distribution is relatively fine, in order to ensure high shovel productivity and crusher throughput.

From a purely production standpoint, fine muck size has distinct advantages. Accompanying this high degree of fragmentation is a high incidence of blast induced damage and potentially higher explosive costs. This damage may not be important during intermediate pushbacks, but will become increasingly so when final walls are excavated. A typical muck pile is pictured in Photograph 3.

Strictly from a blast damage point of view, and not from a production point of view, when blasting takes place with two free faces, the results are significantly improved. This is shown by a comparison of Photograph 4 and Photograph 5.

Blasting under "choked" conditions, resulting from either an adjacent muckpile or solid (unblasted) rock, can have a significant impact upon both the efficiency of the blast and upon the severity of blast damage. The increase in confinement due to lack of burden relief results in excessive periods of gas pressure action. High pressure blast gases jet into and extend both naturally occurring fractures and strain energy created fractures. If these gases are allowed to act for extended periods of time and are not used in the displacement of the fragmented burden, the resulting damage can be severe. This damage not only effects volumes of rock that were meant to be undisturbed but can also disturb ground within the blast pattern, thus increasing the potential for disruption of adjacent charges. By reducing the displacement of the fragmented burdens, muckpile looseness is sacrificed and additional explosive costs are incurred in achieving a high degree of fragmentation to maintain production.

In at least two cases damage extended up to 30 m behind the last row of blastholes. Both blasted patterns were in the South Structural Domain (Figure 5), and the crack patterns that developed were consistent in orientation to major joint sets in this domain. There were, however, marked differences in both the orientation and magnitude of the damage, as well as in the development of the crack patterns. The orientation and development sequence appear, in these two cases, to be related to the initiation sequence of the blast. The preferential displacement along the different joint sets closely follows the geometry of a given initiation time interval (see Figure 12 and Photographs 5, 6 and 7). In some cases the observed displacement was up to 1 m.

The ejection of stemming is often related to the efficiency of the explosive at breaking the rock; it can also be related to the confinement of the blast. Photographs 8 and 9 clearly show the difference between blasting with no free faces and two free faces. With no free faces, the gases produced by the explosives are overly confined, this results in energy being wasted on the high velocity ejection of the stemming material and on the propagation of fractures distant from the blast instead of displacing the burden and providing progressive free face relief.

## 7.0 CONCLUSIONS AND RECOMMENDATIONS

The purpose of this research project was to study the application of an integrated monitoring package to the evaluation of open pit blasting. The study was undertaken over a relatively short time period in a mine where production was of the essence. Hence, some questions still remain to be answered.

### 7.1 BMX Blast Monitor

The results of the program show mixed success with this system while clearly indicating the potential for the system in optimizing open pit blasting.

Successful: The system was successfully used to measure the velocity of detonation (VOD) of the explosive product when the blasthole was advantageously located.

The system provided an indication of the amplitude of ground disturbance due to blasting through signal attenuation (both frequency and amplitude).

Not successful: The system could not identify individual blasthole energy in large (≈ 200 holes) production blasts with relatively close delaying and the degree of ground disturbance.

The system required sensor placement in the near field and a rock mass that is not highly disturbed, neither of which could be successfully achieved at this mine site.

These results suggest that it is not possible to begin optimization using the system in this specific large scale production environment; although there is clearly an application in both a smaller scale environment where initially one or two holes (response calibration and product evaluation - VOD), and subsequently a modest number of holes can be tested, and potentially in a large scale operation with different operating conditions. The application is also good for final pit walls and trim blasts where holes must be fired with distinct time separation to minimize damage.

### 7.2 Videotaping and Photography

The program indicated that use of more readily available 35 mm cameras and camcorders can go a long way towards replacing traditional high speed cameras in providing visual records of blast events. The photography and videotaping provided especially good data on the following blast features:

- Initiation sequence;
- Stemming ejection;
- Blast induced damage;
- Muckpile shape.

## 8.0 OVERALL BLAST OPTIMIZATION PROGRAM

The following represents a possible continuation of research into fully assessing complex blasthole geometries and initiation sequences.

### 8.1 BMX Blast Monitor

The potential of the blast monitor could be further defined through the following steps:

- Monitoring single hole blasts at varying burden distances. This would allow evaluation of the current burden, definition of single hole energy levels and also the time interval of individual energy pulses.

- Successively adding holes to the single hole to establish hole interaction between holes, then within a row and finally between rows. This would establish a database of increasing blast complexity.

- Monitoring of velocity of detonation with varied stemming material (and no stemming) and varied product sleep times to further define stemming and explosive characteristics.

It is also conceivable that the degree of blast induced damage given a specific blast pattern could be established by multi-stationed monitoring of blast vibrations and detailed damage assessment.

## 8.2 Videotaping and Photography

The photographic methods could be further advanced through placing markers within the blast to allow specific hole detonations to be noted. Also, markers of specific orientation and spacing would aid in the interpretation of the pictures.

The SLR photography could also be extended to examine fragmentation when changes to the blast pattern are made during single hole and small blasts.

## REFERENCES

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Golder Associates, "Open Pit Blasting Manual", 1983.

Hagan, T.N., "Designing Blasts for Increased Slope Stability", Proceeding of the 4th International Rock Mechanics Congress, Switzerland, 1979.

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# TABLE 1
# BLAST MONITORING SUMMARY

| Blast Location | No. of Holes | Max. Charge/ 25 ms (kgs) | Planned Duration (ms) | No. of Free Faces | Instrument Distance (m) | Actual Duration (ms) | Peak Recorded Vibrations (mm/s) | Comments |
|---|---|---|---|---|---|---|---|---|
| #3 Drill 1212.5 Bench Feb. 27 (52) | 142 | 5100 (9 holes) | 1120 | 1 | 18 | 933 (a) | B1R B1T B2R B2T<br>134 111 320 170 | Small amounts of fracturing (trending E-W). |
| #7 Drill 1312.5 Bench Feb. 28 (25) | 105 | 4000 (7 holes) | 1045 | 2 | 18 | 691 (a) | B3R B3T B4R B4T<br>243 109 +450 106 | Small amounts of fracturing (trending E-W). |
| #5 Drill 1312.5 Bench Mar. 14 (28) | 133 | 4500 (8 holes) | 940 | 1 | 30 | 1047 | B5R B5T B6R B6T<br>71 Failed 26 56 | Large scale fracturing (trending N-S) at distances of up to 30 m behind last row of blastholes. |
| | | | | | | | | NOTE: Velocities are not vector sums and only represent the velocity in a given direction and hence are minimum values. |

(a) instruments destroyed.

B5R - Denotes instrument number.

# TABLE 2
# VOD SUMMARY

| Blast | | VOD Probe | Average Velocity (m/s) | Comments |
|---|---|---|---|---|
| 21.2.90 #4 Drill | 1100 Bench | 1 | — | Ground Displacement |
| | | 2 | — | Ground Displacement |
| 23.2.90 | | 3 | — | Ground Displacement |
| | | 4 | — | Ground Displacement |
| 15.3.90 | | 5 | 5500 | Only 2.5 m of blasthole recorded; both probes, same hole |
| | | 6 | 5500 | |
| 16.3.90 | | 7 | 3500 | |
| | | 8 | 3800 | |
| 20.3.90 | | 9 | — | Instrument |
| | | 10 | — | Displaced |

A typical measurement configuration and results is presented in Figure 11.

# TABLE 3
# HIGHLAND VALLEY COPPER VOD MEASUREMENTS
# IRECO EMULSION HD 30% PRILLS

VOD7

| Distance | Time | Velocity |
|---|---|---|
| 0 | NA | |
| 0.5 | 194.165 | NA |
| 1 | 194.4 | 2128 |
| 1.5 | 194.54 | 3571 |
| 2 | 194.61 | 7143 |
| 2.5 | 194.735 | 4000 |
| 3 | 194.89 | 3226 |
| 3.5 | 195.29 | 1250 |
| 4 | 196.18 | 562 |
| 4.5 | 198.03 | 270 |

VOD8

| Distance | Time | Velocity |
|---|---|---|
| 0 | NA | |
| 0.5 | 0.405 | NA |
| 1 | NA | NA |
| 1.5 | 0.71 | 3279 |
| 2 | 0.85 | 3571 |
| 2.5 | 0.925 | 6667 |
| 3 | 1.06 | 3704 |
| 3.5 | 1.355 | 1695 |
| 4 | 2.265 | 549 |
| 4.5 | 2.53 | 1887 |

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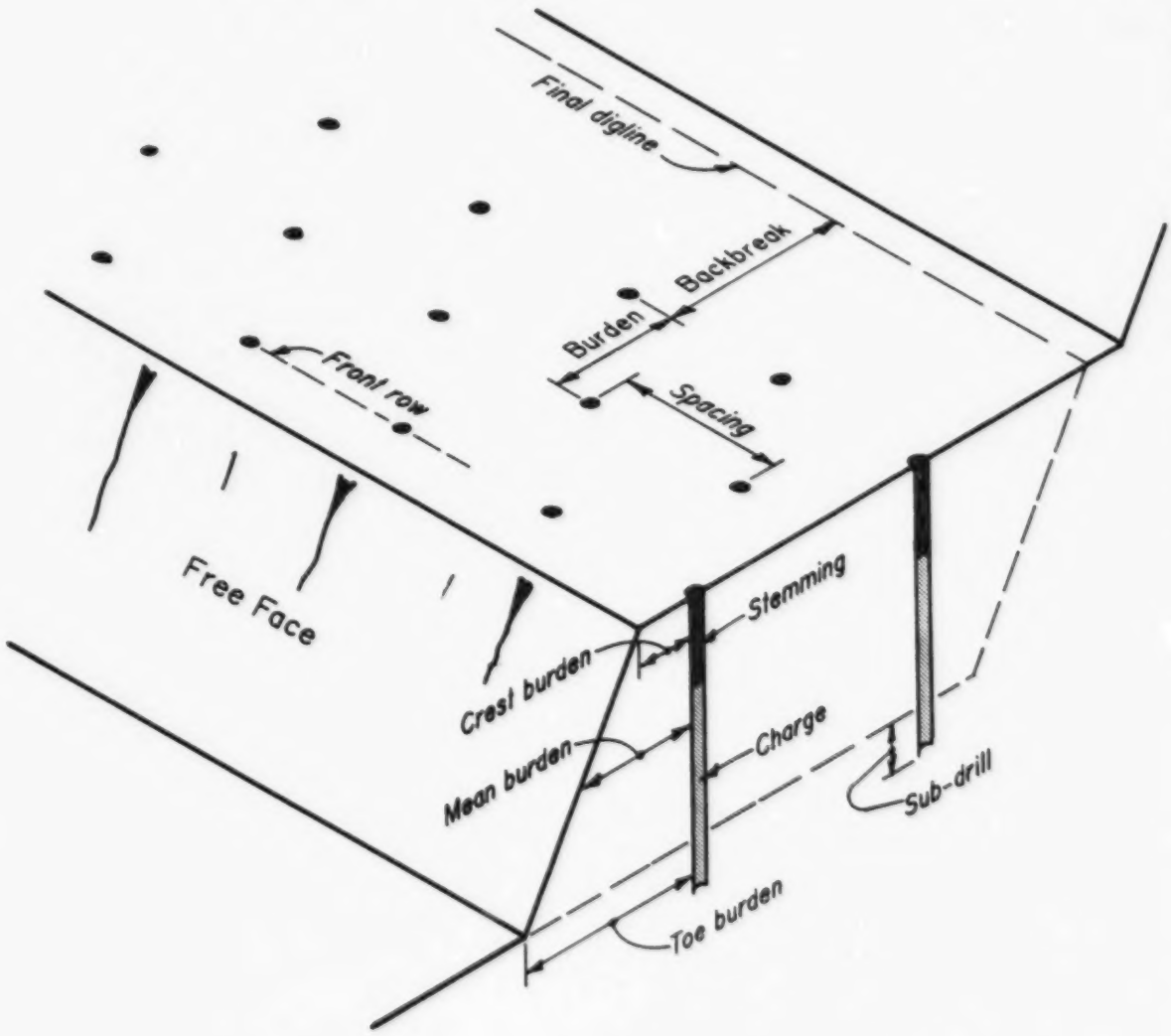
## TABLE 4
## BLASTING PRACTICES AT OTHER HARD ROCK MINES

| Hole Diameter | Burden | Spacing | Bench Height | Subgrade | Stemming | Charge/ Hole | Powder Factor | Initiation Sequence | Size of Blast |
|---|---|---|---|---|---|---|---|---|---|
| 10 5/8"<br>270 mm | 26-30'<br>7.8-9.1 m | 30-34'<br>9.0-10.4 m | 41'<br>12.5 m | 5'<br>1.5 m | 21-23'<br>6.4-7 m | 800-1200 lbs.<br>364-546 kgs<br>Emulsion | 0.4 lbs./ton<br>0.2 kg/tonne | Variable | 80-220 |
| 9 7/8"<br>251 mm | 22'-25'<br>6.7-7.6 m | 22'-25'<br>6.7-7.6 m | 40'<br>12.2 m | 3'-5'<br>0.9-1.5 m | No specs | 720-960 lbs.<br>327-436 kgs<br>Slurry | 0.33-0.47 lbs./ton<br>0.17-0.24 kg/tonne | V from middle | 40 |
| 9 1/8"<br>232 mm | 20'<br>6.1 m | 24'<br>7.3 m | 33'<br>10 m | 5'<br>1.5 m | 22'<br>6.7 m | 500 lbs.<br>227 kgs<br>Watergel | 0.45 lb./ton<br>0.23 kg/tonne | En echelon from end | 40 |
| 8"<br>203 mm | 15'<br>4.6 m | 20'<br>6.1 m | 25'<br>7.6 m | 6'<br>1.8 m | 8'-10'<br>2.4-3 m | 300-400 lbs.<br>136-182 kgs<br>ANFO | 0.4 lb./ton<br>0.2 kg/tonne | V-1 from middle | 100 |
| 9"<br>229 mm | 16'<br>5 m | 20'<br>6 m | 16'<br>5 m<br>150 | 5'<br>1.5 m | No specs | 150 - 180 lbs.<br>68-81 kgs<br>Anfo<br>Slurry | 0.36 lbs./ton<br>0.18 kg/tonne | Row by row | 160 |
| 9 7/8" | 20'<br>6.1 m | 23'<br>7 m | 30'<br>9.6 m | 7'<br>2.1 m | 25'<br>7.6 m | 590 - 840 lbs.<br>ANFO<br>Superan<br>Emulsion | 0.47 lb./ton<br>0.24 kg/tonne | Row by row | 40 |
| 12 1/4" | 24<br>7.3 m | 28<br>8.5 m | 30'<br>9.6 m | 7<br>2.1 m | 25<br>7.6 m | 930 - 1290 lbs.<br>ANFO<br>Superan<br>Emulsion | 0.49 lb./ton<br>0.24 kg/tonne | Row by row | 40 |

## BLAST DESIGN VARIABLES

Figure 1




***SCHEMATIC ONLY***
***Not to Scale***

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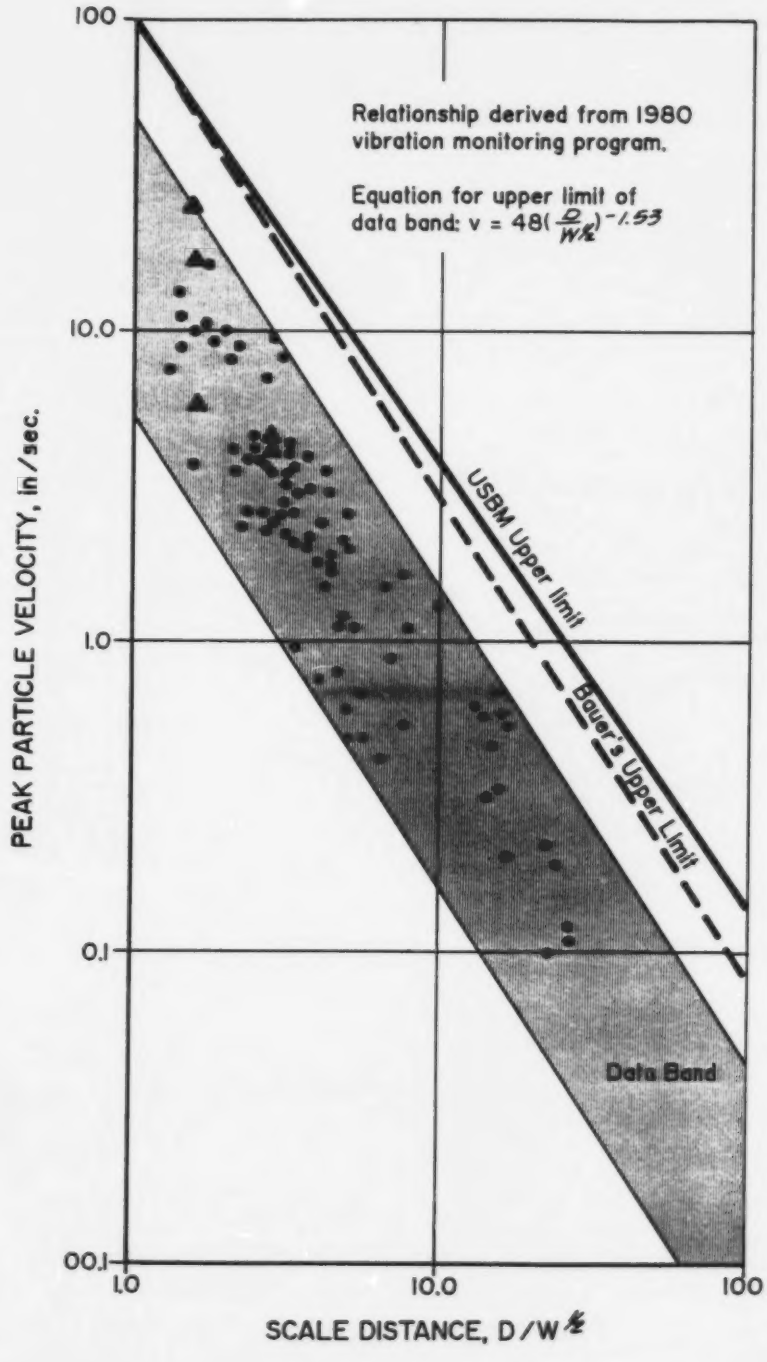
# TYPICAL PEAK PARTICLE VELOCITY Vs SCALED DISTANCE

Figure 2

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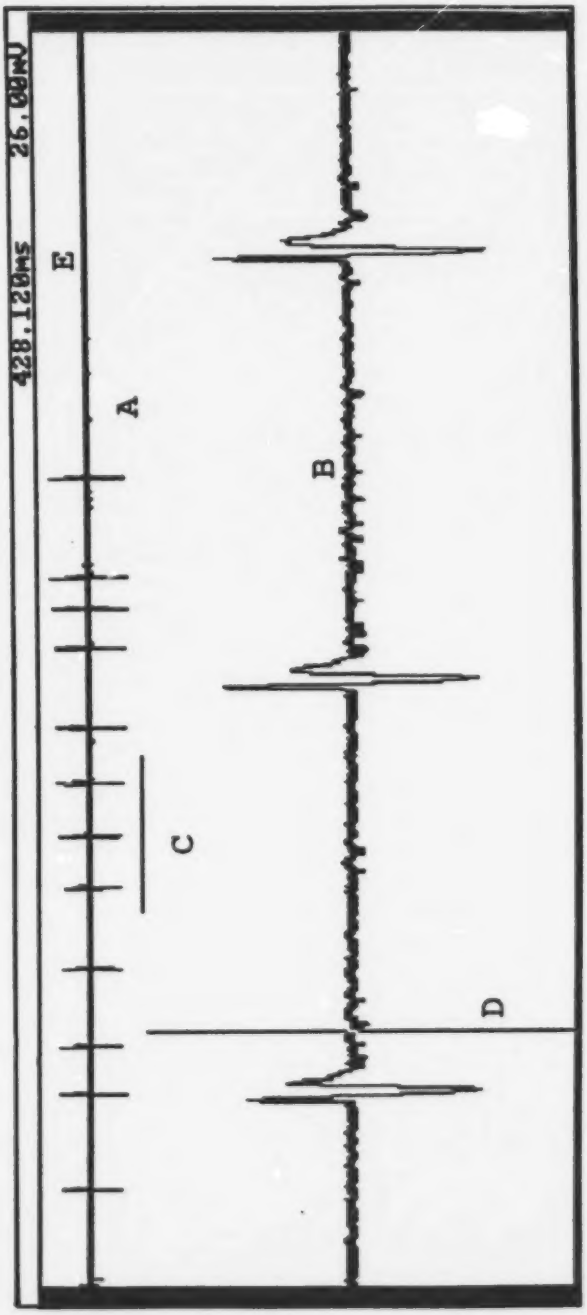
**TYPICAL BMX VIBRATION WAVEFORM**

Figure **3**

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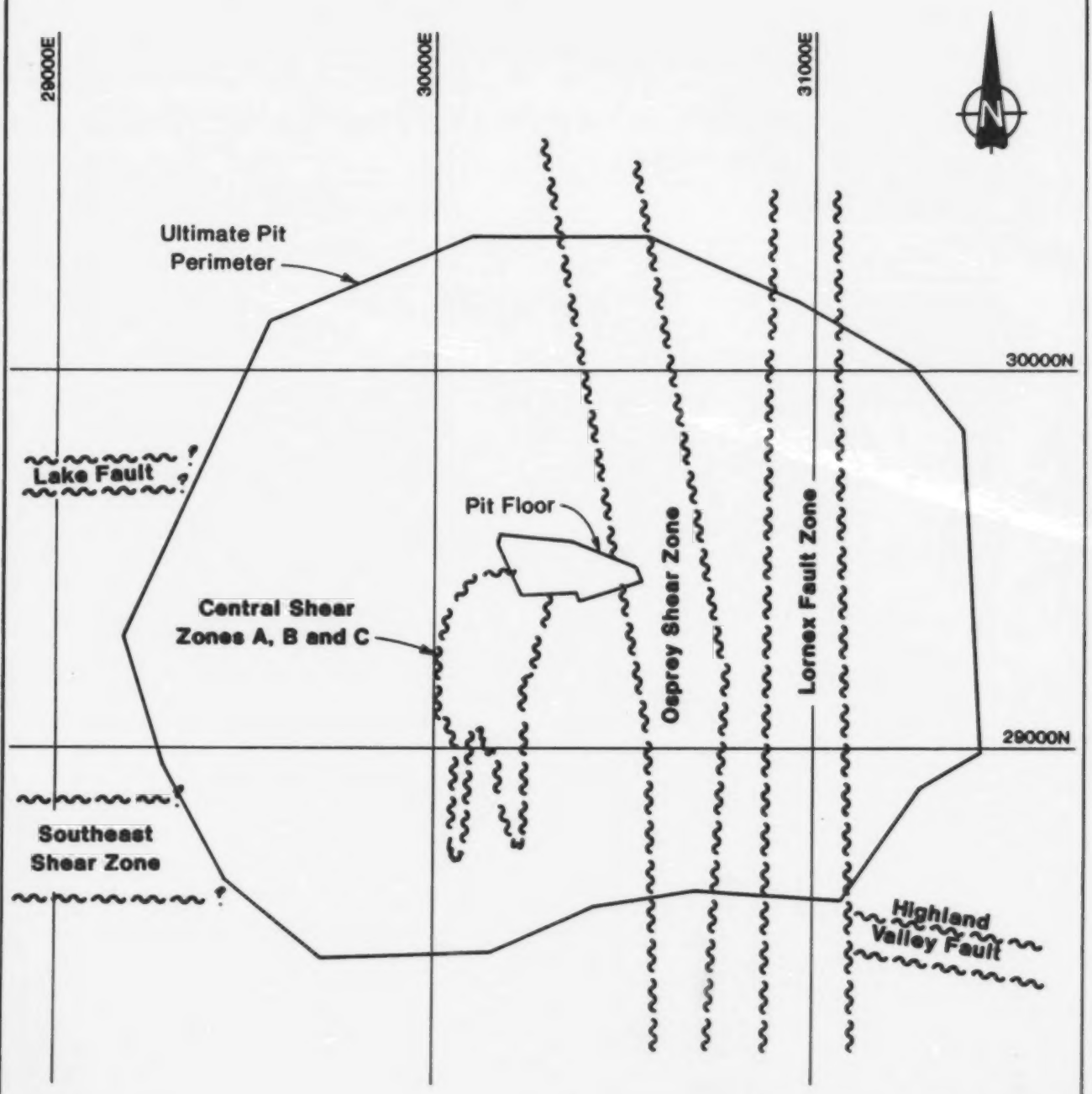
# PLAN OF THE GEOLOGY OF THE VALLEY PIT

Figure 4




Scale 1 : 15000

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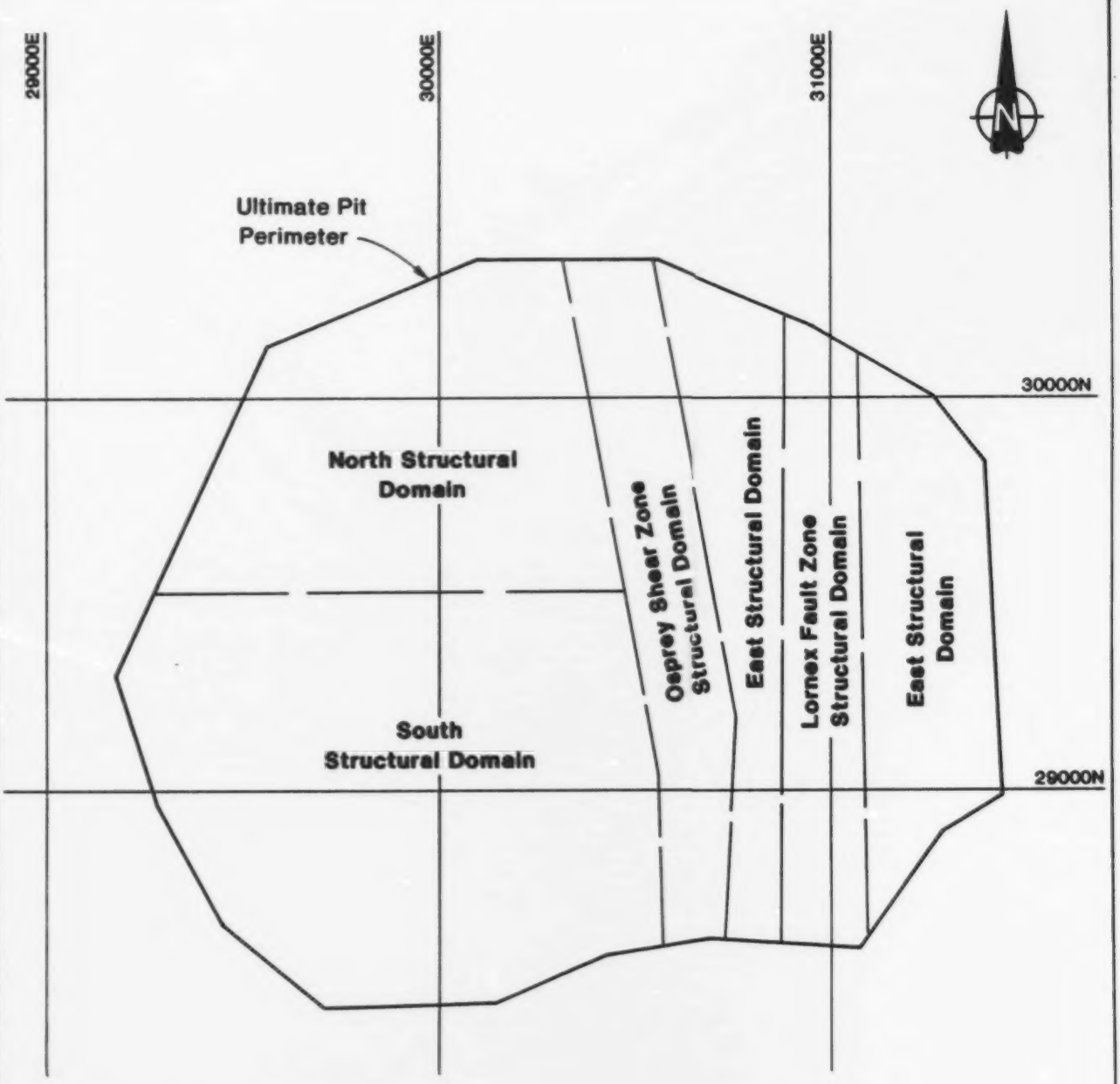
# PLAN OF STRUCTURAL DOMAINS OF THE ULTIMATE VALLEY PIT

Figure 5




Scale 1 : 15000

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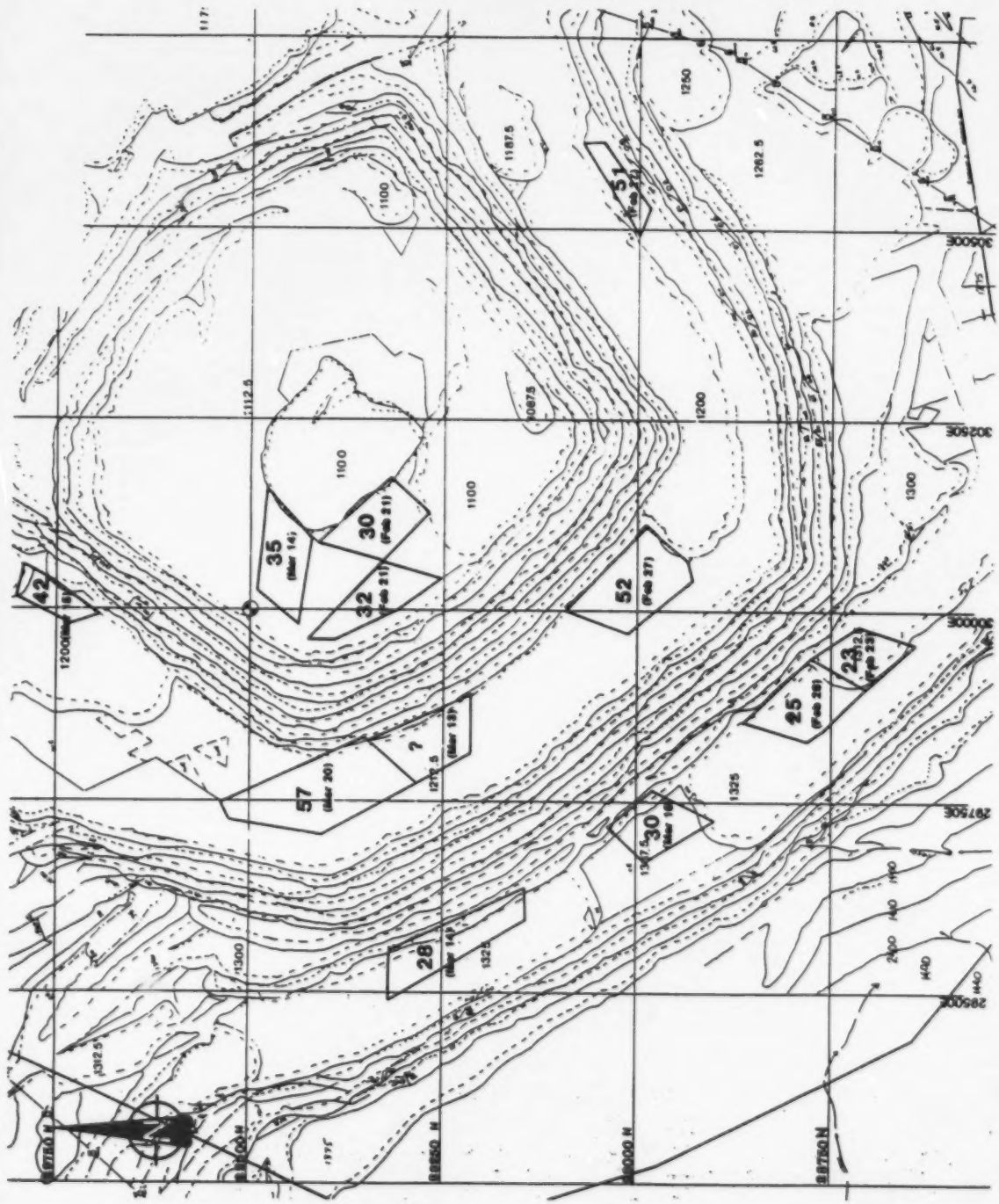
# APPROXIMATE BLAST LOCATIONS
# VALLEY PIT – March 1, 1990

Figure 6

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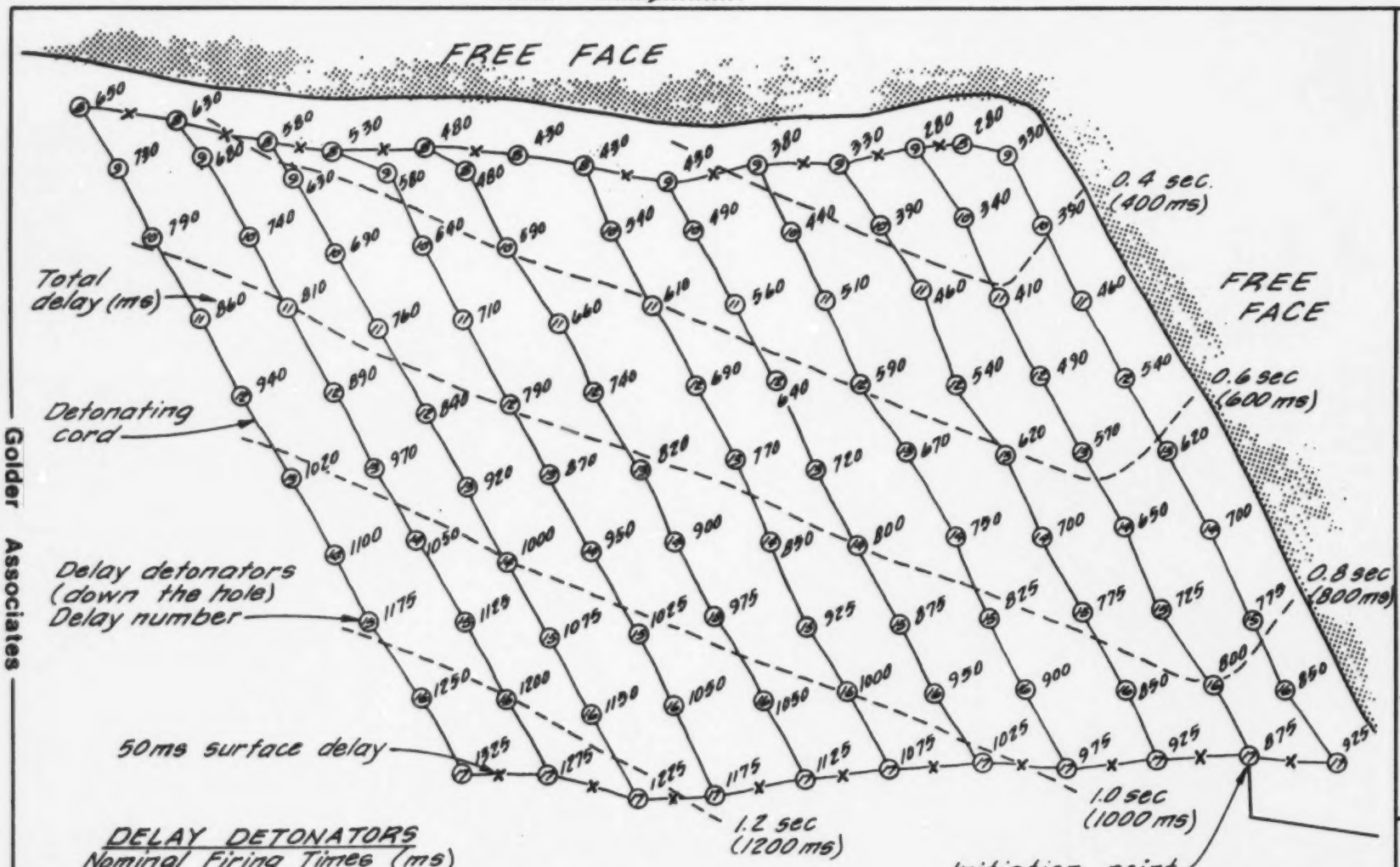
PROJECT NO. 902-1410 DRAWN V.T. REVIEWED B.F. DATE May '90




## DELAY DETONATORS
Nominal Firing Times (ms)

| Delay No. | | Delay No. | |
|---|---|---|---|
| 8 | 230 | 13 | 570 |
| 9 | 280 | 14 | 650 |
| 10 | 340 | 15 | 725 |
| 11 | 410 | 16 | 800 |
| 12 | 490 | 17 | 875 |

Not to Scale

Blast No. 25 (see Figure 6 for location)

# TYPICAL SEQUENCING WHEN FIRING TO TWO FREE FACES

Figure 7

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# TYPICAL SEQUENCING WHEN FIRING TO ONE FREE FACE

Figure 8




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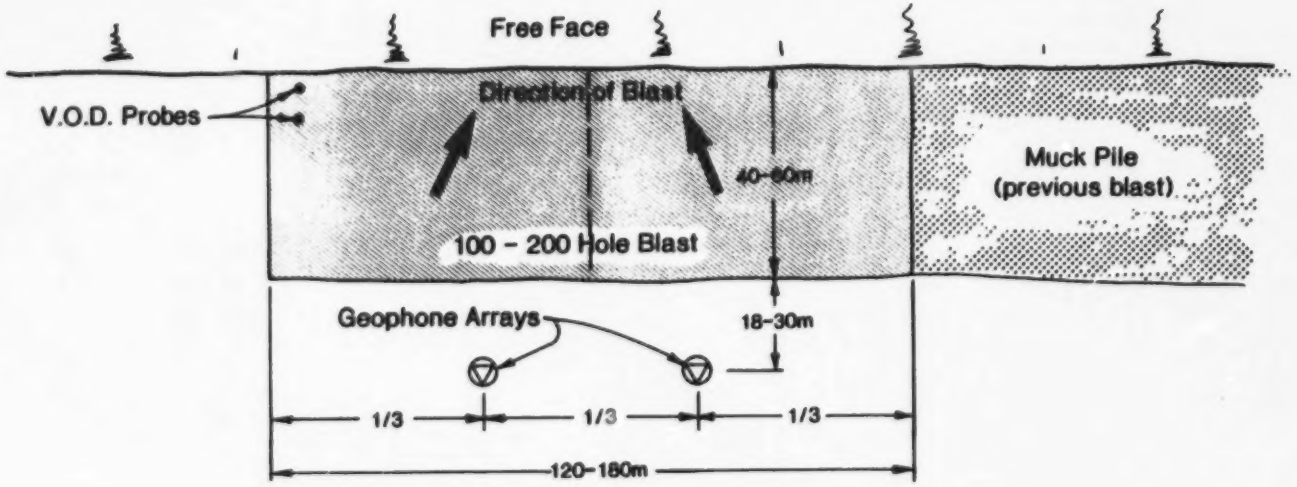


8mm Video

Located at a Distance of
Approximately 400-1000m
Horizontal and 60-120m Vertical

35mm Camera

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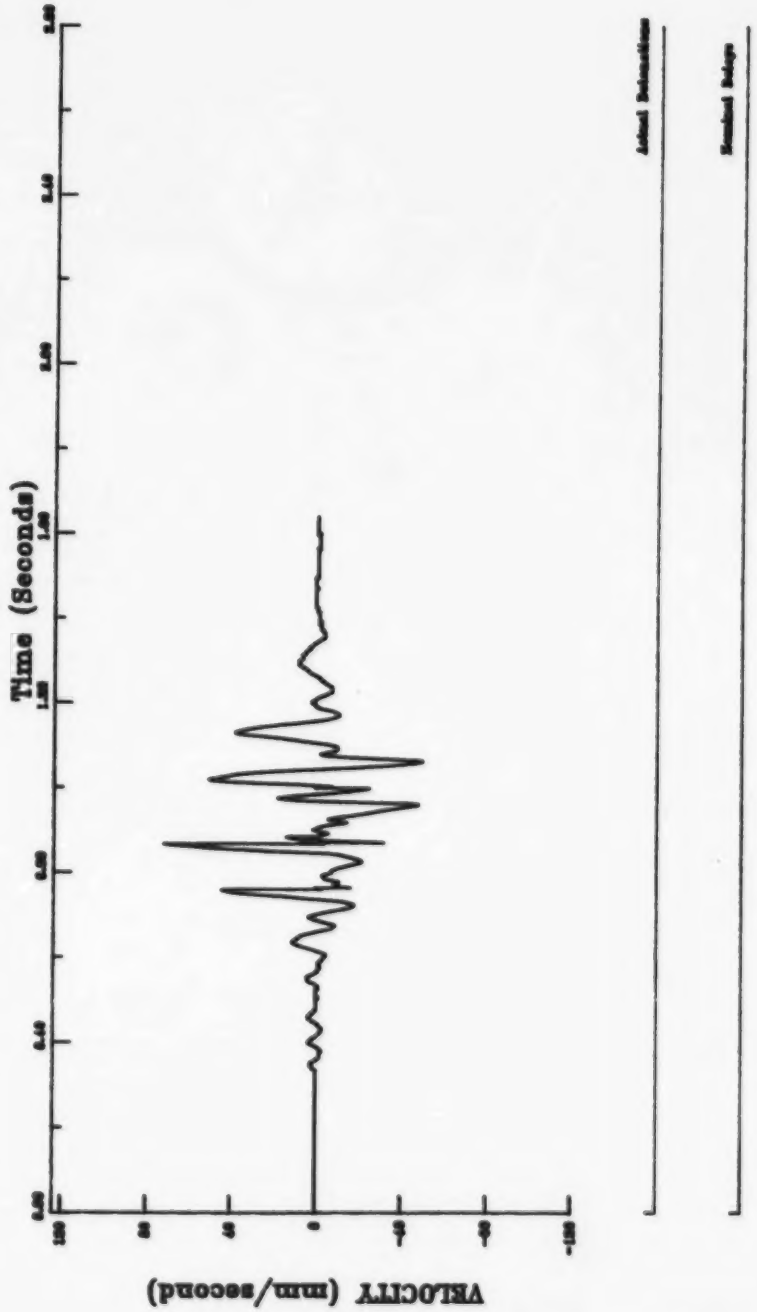
TYPICAL MONITORING CONFIGURATION

Figure 9

**TYPICAL BMX VIBRATION WAVEFORM FROM THIS SITE**

Figure **10**

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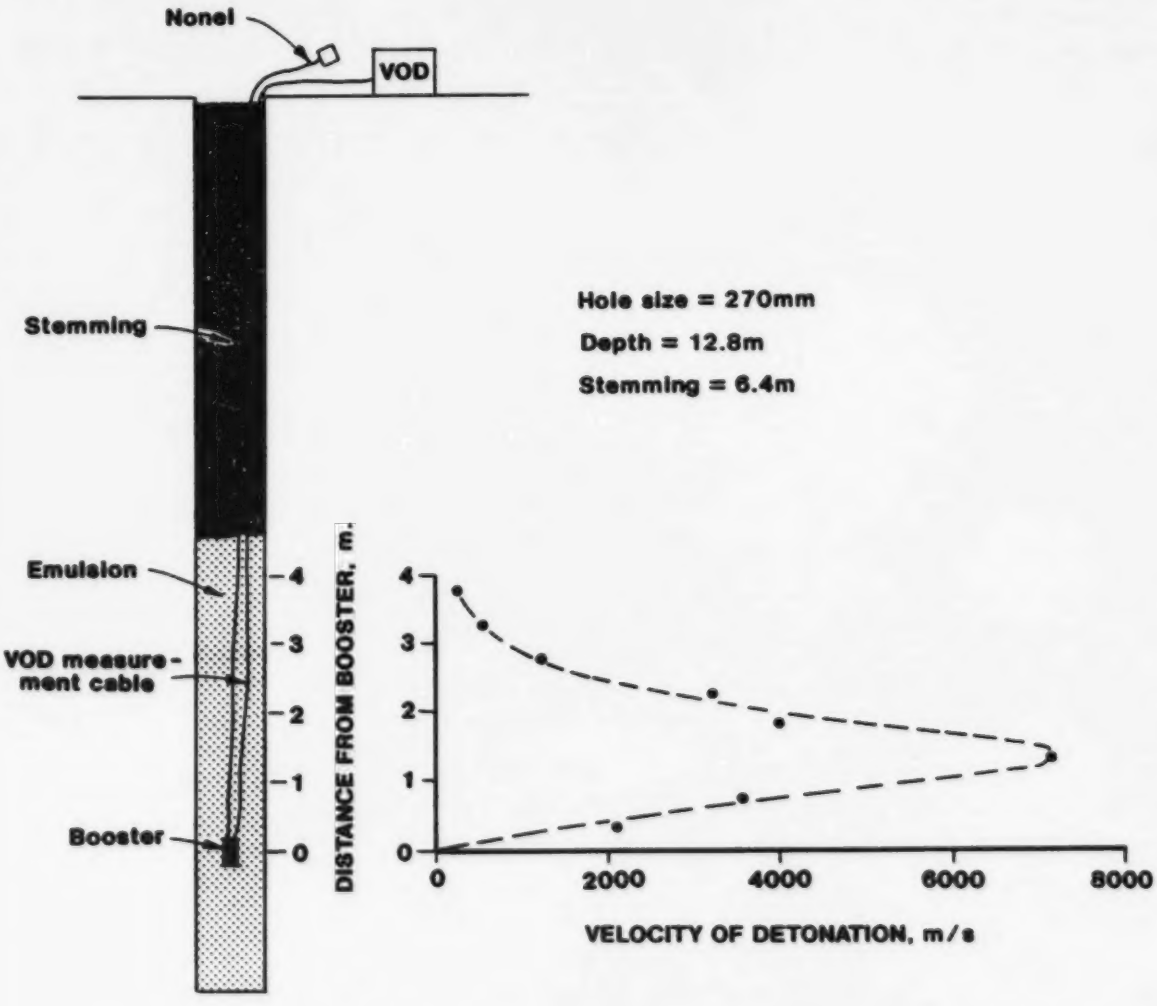
# TYPICAL VOD MEASUREMENT

Figure 11

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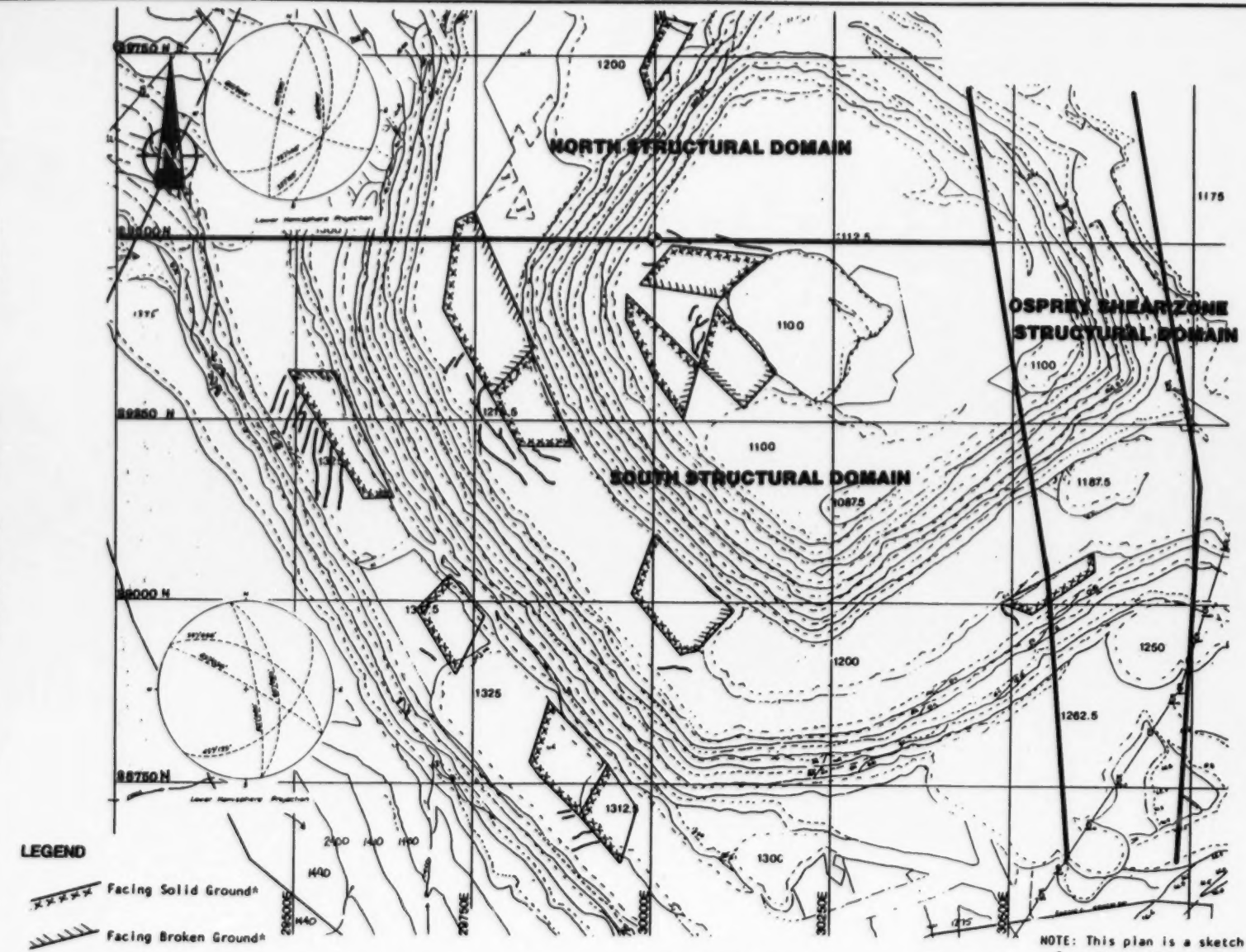
PROJECT NO. 902/410 DRAWN – REVIEWED A.F. DATE May 1990

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BLAST INDUCED DAMAGE
VALLEY PIT – March 1, 1990

Figure 12

# APPENDIX I

# BLAST MONITOR SPECIFICATIONS

## *BLASTRONICS BMX BLAST MONITOR SPECIFICATIONS*

**DATA CHANNELS** From 1 to 8 (factory set as per order)

Resolution 8 Bits (1 in 256 or 0.5%)

Sampling

| Frequency | Interval | Record Time (sec.) |
|---|---|---|
| 200 kHz | 5µsec | 0.33 |
| 100 kHz | 10µsec | 0.66 |
| 50 kHz | 20µsec | 1.3 |
| 10 kHz | 100µsec | 6.6 |
| 5 kHz | 200µsec | 13.2 |
| 2 kHz | 500µsec | 32.8 |
| 1 kHz | 1 msec | 65.6 |

Input Voltage 8 ranges: 0.3, 0.6, 1.2, 2.4, 4.8, 9.6, 19.2, 38.4 Volts (corresponds to 10, 20, 40, 80, 160, 320, 640, 1280 mm/s for re-usable geophone transducer supplied with unit). Ranges for other sensors depend on sensor sensitivity.

Triggering 14 Levels of Threshold trigger on each channel, or single Wirebreak trigger.

**BATTERIES** 2 x 6v integral, rechargeable gel cells, 2.6 Ahrs 12 hours operation from fully charged (24 hr option available). Estimated minimum batterly life 3 years in normal service. External DC connection facility for extended operating time.

**RESIDENT SOFTWARE**

(Menu system)

**Edit:** document a record, set amplifier gains, triggering method and sensor calibration.

**Initialise:** prepare system for data capture.

**Disk storage:** Download data to floppy disk.

**Enter:** input nominal delays for comparison with actual delays on vibration trace.

**View/Analyse:** display vibration waveforms on screen and tabulate delay times and vibration levels for each charge. Plot or compare traces from different blast records. Output to optional printer.

**DIMENSIONS** 440 mm (l) x 385 mm (w) x 212 mm (h) excluding handle.

**WEIGHT** 15 kg

**INTERNAL COMPUTER** Toshiba T 1100 Plus, 7.16 MHz, 640 kB RAM, dual 720 kB disk drives, IBM compatible, battery powered, RS-232, RGB Colour Video and composite CRT ports.

**OPTIONS** External graphics mode printer: Epson MX80 and compatibles, IBM ProPrinter and compatibles. Other printers may be supported - please contact supplier for details.

## GEOSPACE CORPORATION
## GEOPHONE SPECIFICATIONS

MODEL - GS-20DX 8 Hz 395 Ohms

| Description | Specification | Tolerance |
|---|---|---|
| Natural frequency (Fn) | 8 Hz | +/- 0.5 Hz |
| Tilt angle | +/- 5 degrees | |
| Frequency tolerance at tilt | 8 Hz | +/- 0.8 Hz |
| Coil resistance @ 25° C (Rc) | 395 Ohms | +/- 5% |
| Intrinsic voltage sensitivity (G) | 0.276 V/cm/sec | +/0 10% |
| Normalized transduction constant | 0.352 | Rc |
| Open circuit damping (Bo) | 0.39 | +/-10% |
| Damping constant (BcRt) | 687 | |
| Moving mass | 11 g | +/- 0.5 g |
| Coil excursion P-P | >0.081 cm | |
| Operating temperatures | -34° to 71° C | |
| Storage temperatures | -51° to 85° C | |

Dimensions:

| | |
|---|---|
| Height | 3.35 cm |
| Diameter | 2.54 cm |
| Weight | 87.3 g |

**APPENDIX II**

**VISUAL RECORDING**

# Canon EOS 630

## Specifications

**Type:**
35 mm autofocus, single-lens reflex camera with electronically-controlled automatic exposure, focal plane shutter, and built-in motor drive.
**Format:** 24 × 36 mm
**Usable lenses:**
Canon EF lenses (full aperture metering only)
**Standard Lenses:** EF 50 mm f/1.8
EF 35-70 mm f/3.5-4.5
**Lens Mount:**
Canon EF Mount (full electronic signal transfer system)
**Viewfinder:**
Fixed eye-level pentaprism. Gives 94% vertical and horizontal coverage of actual picture area, and 0.8 × magnification at infinity with a standard 50 mm lenses.
**Dioptric Adjustment:**
Built-in eyepiece is adjusted to standard −1 diopter. (eyepoint: 19.3 mm)
**Focusing Screen:**
Overall new laser-matte/AF frame (type C). Six types of interchangeable screens are available optionally.



**Mirror:**
Quick-return half-mirror with shock and noise absorber.
**Viewfinder Information:**
Displayed at the bottom of the viewing area.
(1) 7-segment LCD digit and character display
1. Shutter speed—flashes at 2Hz to give out-of-coupling range warning.
2. Aperture value—flashes at 2Hz to give out-of-coupling range warning.
3. Metered manual exposure level—OP, oo, CL
4. Depth of field AE—dEP 1, dEP 2

(2) LCD mask character display
1. ⋈ —AE lock indicator in partial metering mode.
2. M—Manual exposure indicator
3. ϟ —Flash charge completion indicator
4. +/−—Exposure compensation indicator
5. ●—AF in-focus indicator (flashes at 6Hz when AF is not possible.)

**Light Metering System:**
TTL full aperture metering using SPC. Two selectable metering patterns: evaluative metering and partial metering (approx. 6.5% of the picture area). Stopped-down metering is not possible.

**Shooting Modes:**

1. Green mark standard shooting mode with P.I.C. (Programmed Image Control)
2. Shutter-Priority AE
3. Aperture-Priority AE
4. Intelligent Program AE with variable shift function
5. Depth of Field AE
6. Manual
7. Flash AE (A-TTL program flash AE and TTL program flash AE with P.I.C. #7 or specified Canon Speedlites)

**Camera-shake Warning:**
Operates for program AE, aperture-priority AE, and depth-of-field AE modes. When automatically set shutter speed falls 0 to 0.5 steps below 1/focal length of the lens in use, the electronic beeper sounds. Beeper can also be canceled.
**Metering Range:**
EV 1-20 (EV −1 to 20 in normal temperature)
Conversion with 50 mm f/1.4 at ISO 100.
**Film speed:**
ISO 25-5000 is automatically set by 1/3 step according to DX code standard. ISO 6-6400 can also be set manually.
**Exposure Compensation:**
±5 steps by 1/2-step increment
**Auto Exposure Bracketing:**
±5 steps in 1/2-step increment. Three continuous exposures are taken in sequence of underexposure, correct exposure according to camera's meter and overexposure.
**Multiple Exposures:**
Presetting up to nine exposures is possible. Automatically cleared upon completion.
**AF Control System:**
TTL-SIR (TTL secondary imaged registration) phase-detection type using BASIS (base-stored image sensor). AF operation starts when the shutter button is pressed halfway. AF in-focus indicator lights upon ranging completion.
**Three selectable modes:**
1. ONE SHOT: AF operation ends and focus is locked once ranging is completed. Shutter does not release until ranging is completed.
2. AI SERVO: Focus continuously adjusts to follow the subject movement. Shutter can be released at any time regardless of ranging completion. Equipped with focus prediction.

3. Manual: By rotating the manual focusing ring after focus mode switch is set to "M".
   AF Working Range:
   EV 1-18 at ISO 100
   AF Auxiliary Light:
   The near infrared light (peak sensitivity: 700 nm) is automatically projected with specified Canon Speedlites.

**Shutter:**
Vertical-travel focal plane shutter with soft-touch electromagnetic release. All speeds electronically controlled.

**Shutter Speed:**
1/2000–30 secs and bulb. X-sync is 1/125 sec. Can be set in 1/2-step increments.

**Self-timer:**
Electronically controlled with a delay of approx. 10 secs indicated by blinking LED operation confirmation lamp.

**Film Loading:**
After film positioning and back cover closure, the film automatically advances to the first usable frame and then stops (approx. 1.0 sec)

**Film Wind:**
Automatic using the built-in miniature motor. Confirmation by the film transport bar marks in the display panel.



**Film winding Mode:**
Two selectable modes; S (single exposure) and C (continuous exposure at the maximum speed of approx. 5 frames/sec. with ONE SHOT AF mode and approx 2.5 frames/sec. with AI SERVO AF mode).

**Film Rewind:**
Automatic using the built-in miniature motor. Starts when the film end is reached and then stops (with 24-exp. film, approx. 4 secs.). Mid-roll rewind performed by pressing the film rewind button.

**Flash Contact:**
Coupled directly to the camera by X-sync contact on the accessory shoe.
Automatic flash (using the Speedlite 420EZ/300EZ with the camera set to "P"):
A-TTL flash auto—Using the camera's A-TTL program and the flash's near-infrared preflash, the correct aperture value is automatically set. X-sync speed is also automatically set between 1/60 and 1/125 sec. upon flash charge completion. TTL control system which meters the light reflected from the film surface. Automatic fill-in flash is possible.

**Grip:**
Interchangeable
The GR30 (without remote control terminal): standard.
The GR20 with remote control terminal: available optionally.
The large-size, Grip GR10 is available optionally.

**Depth-of-Field Check:**
By pressing the depth-of-field check button.

**LCD Display Panel:**
Displays only the information required at the time, e.g. shooting mode, film winding mode, AF mode, shutter speed, aperture value, film speed, or battery check. Timer funciton for 6 secs is provided; the display is held on for 6 secs; after the switch i.e., shutter button is pressed. Can be illuminated by electro luminescence. Custom Functions: Built-in, seven kinds.

**Power Source:**
One, six-volt lithium battery pack (2CR5). Battery replaced by removing the grip.

**Battery Check:**
By pressing the battery check button. Three energy levels are shown by the battery check bar marks in the display panel.

**Back Cover:**
Interchangeable. Opened by sliding the latch with safety lock. Quartz Date can be attached.

**Dimensions:**
148 (W) × 108.3 (H) × 67.5 (D) mm
(5-13/16" × 4-1/4" × 2-5/8")

**Weight:**
670g (23.4 oz.) body only.
(All data are based on Canon's Standard Test Method).

Subject to change without notice.

# Specifications

## H660/E640

| | |
|---|---|
| **Power supply (rated):** | 6 V DC |
| **Power consumption:** | 8.3 W (H660); 8.8 W (E640) |
| **Television system:** | EIA standard (525 lines, 60 fields)<br>NTSC color signal |
| **Video recording system:** | 2 rotary heads, helical scanning system<br>Luminance signal: FM azimuth recording<br>Color signal: Converted subcarrier phase invert recording<br>(8mm/Hi8 video standard) |
| **Audio recording system:** | Frequency multiplexing with the video signal by two rotary heads (one channel) |
| **Image sensor:** | 1/2 inch CCD (Charge Coupled Device) |
| **Tape format:** | 8mm, metal evaporated/metal particle |
| **Tape speed:** | 9/16 ips (14.345 mm/s) |
| **Maximum recording time:** | 2 hours with P6-120/E6-120 videocassette |
| **Lens:** | (H660) f/1.4, 6X power zoom, 9—54 mm focal length, macro mechanism (wide)<br>(E640) f/1.4, 8X power zoom, 8.5—68 mm focal length, macro mechanism (wide) |
| **Focusing system:** | Active autofocus, manual focus possible |
| **Minimum auto focusing distance:** | 2 feet (0.6 m) or 4 mm in wide macro |
| **Minimum illumination:** | 7 lux |
| **Filter diameter:** | 46 mm |
| **Electronic viewfinder:** | 0.7-inch monochrome CRT |
| **Microphone:** | Non-directional electret condenser microphone |
| **Input level:** | Video: 1 Vp-p/75 ohms, unbalanced (video terminal);<br>Audio: −10 dBV/more than 47 kohms, unbalanced (audio terminal); −64 dBV/600 ohms, unbalanced (external mic) |
| **Output level:** | (H660) Video: 1 Vp-p/75 ohms, unbalanced (video terminal), 1 Vp-p (Y signal)/0.286 Vp-p (C signal) (S-video output); Audio: −10 dBV/less than 3 kohms, unbalanced (audio terminal)<br>(E640) Video: 1 Vp-p/75 ohms, unbalanced (video terminal); Audio: −10 dBV/less than 3 kohms, unbalanced (audio terminal) |
| **Dimensions: (Width × Depth × Height)** | (H660) 4.9 × 11.7 × 4.8 inches (125 × 298 × 122 mm)<br>(E640) 4.7 × 11.8 × 4.8 inches (120 × 301 × 122 mm) |
| **Weight:** | (H660) 2.9 lb. (1.3 kg) without Battery Pack<br>(E640) 2.8 lb. (1.2 kg) without Battery Pack |

## RF Unit RU-E3

| | |
|---|---|
| **Power source:** | 5 V DC |
| **Power consumption:** | 20 mA (max.) |
| **RF output:** | 3CH/4CH selectable |
| **Dimensions: (Width × Depth × Height)** | 1-7/8 × 2-3/16 × 1-5/16 inches<br>(48 × 56 × 33.5 mm) |
| **Weight:** | 0.12 lb. (55 g) |

Weight and dimensions are approximate.
Errors and omissions excepted.
Subject to change without notice.

CANON IS A REGISTERED TRADEMARK OF CANON INC.

**APPENDIX III**

**PHOTOGRAPHS**

## LIST OF PHOTOGRAPHS

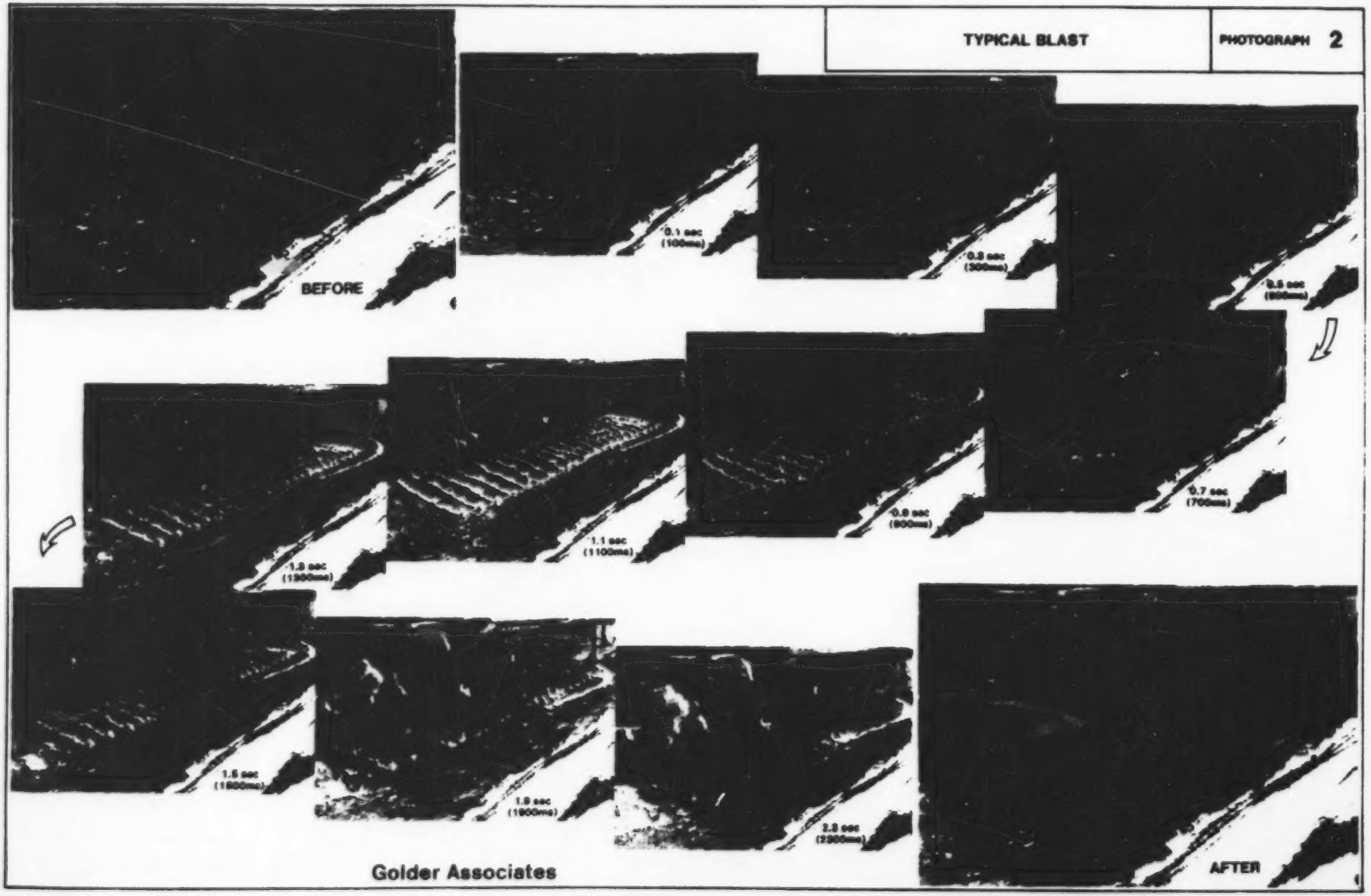

| | |
|---|---|
| Photograph 1 | Valley Pit |
| Photograph 2 | Typical Blast Sequence |
| Photograph 3 | Typical Fragmentation |
| Photograph 4 | Typical Blast to Free Face |
| Photograph 5 | Blast Induced Damage |
| Photograph 6 | Blast Induced Damage |
| Photograph 7 | Blast Induced Damage |
| Photograph 8 | Stemming Ejection - Free Face |
| Photograph 9 | Stemming Ejection - Confined |

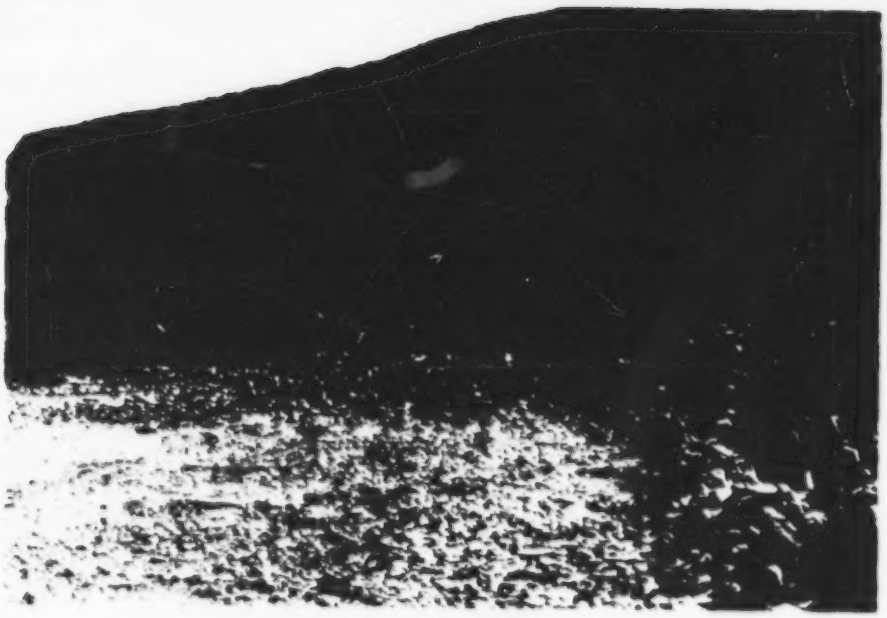
| HIGHLAND VALLEY COPPER–VALLEY PIT | PHOTOGRAPH 1 |
|---|---|

VIEW LOOKING SOUTH–WEST (March 1990)

VIEW LOOKING NORTH–EAST (March 1990)

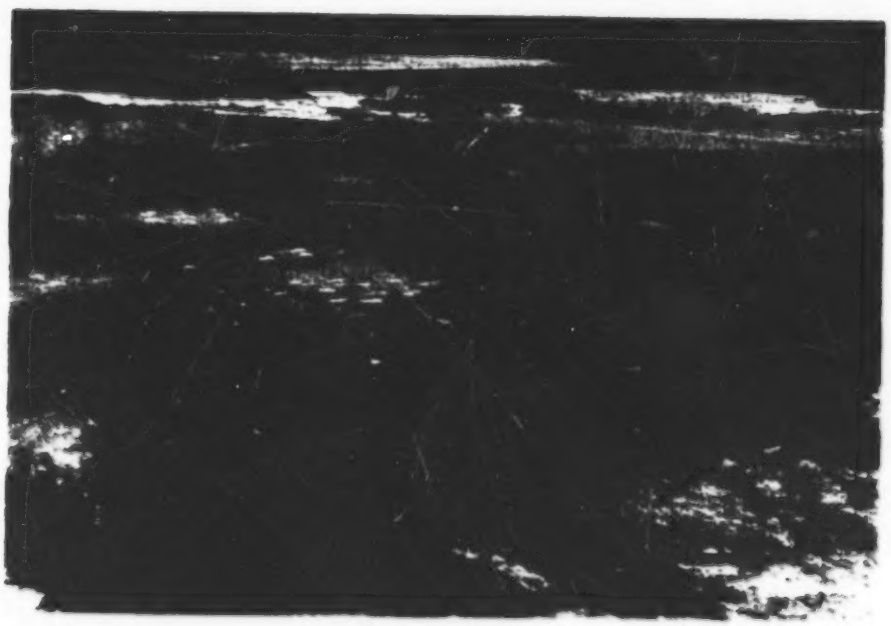
Golder Associates


TYPICAL BLAST
PHOTOGRAPH 2
BEFORE
0.1 sec (100ms)
0.3 sec (300ms)
0.5 sec (500ms)
0.7 sec (700ms)
0.8 sec (800ms)
1.1 sec (1100ms)
1.3 sec (1300ms)
1.6 sec (1600ms)
1.9 sec (1900ms)
2.3 sec (2300ms)
AFTER
Golder Associates



**PHOTOGRAPH 3**

View looking East on 1200 bench (March 14, 1990). Note the relatively small fragment size.



**PHOTOGRAPH 4**

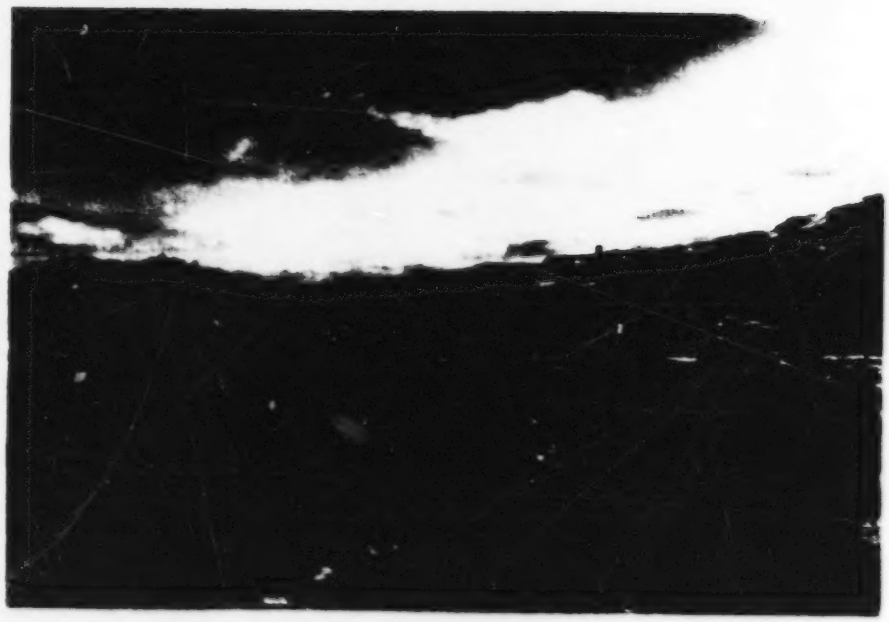
View looking North onto 1312.5 bench (March 16, 1990). This picture shows the results of a blast with two free faces. Lack of backbreak and flyrock can be seen as well as the excellent fragmentation and muckpile shape.

**PHOTOGRAPH 5**

View looking Southeast onto 1325 bench (March 14, 1990). This picture shows the results of a blast with one free face. Extensive fracturing can be seen (in excess of 30 m) along geologic features, extending radially from the blast area.

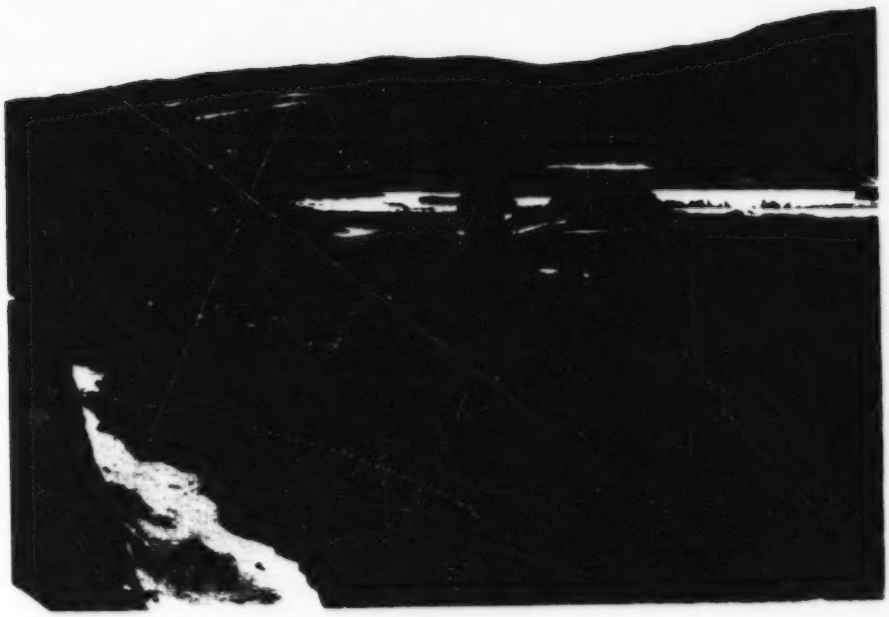
**PHOTOGRAPH 6**

View looking North onto 1212.5 bench (March 13, 1990). Again, the results of a blast with one free face. Extensive fracturing can be seen circumferentially around the area of the blast. These are influenced by geologic features.



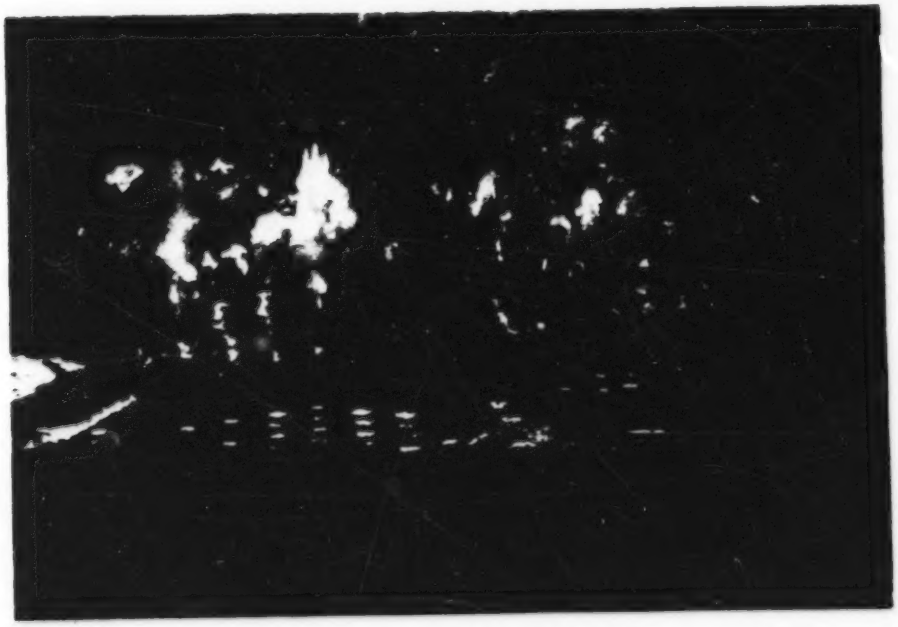
**PHOTOGRAPH 7**

View looking Southeast on 1325 bench (March 14, 1990). An example of the fracturing seen in Photograph 3. The pink flagging (located between the two wooden stands on the right hand side of the picture) is 30 m from the nearest blasthole.



**PHOTOGRAPH 8**

View looking northwest onto 1312.5 bench (February 28, 1990). This photograph (taken 1.4 seconds after ignition) shows the minimal amount of stemming ejection when two free faces are present.



**PHOTOGRAPH 9**

View looking southwest onto 1112.5 bench (March 14, 1990). This photograph (also taken 1.4 seconds after ignition) shows the large amount of stemming material ejected when no free faces are present.

# APPENDIX IV

# BLASTING PRACTICE QUESTIONNAIRE

# PRODUCTION BLASTING SUMMARY

(PAGE 1 OF 2)

PROPERTY: _____

LOCATION: _____

## DESIGN PARAMETERS

HOLE SIZE(S): _____ BENCH HEIGHT: _____

BURDEN(S): _____ SUBGRADE: _____

_____ STEMMING: _____

SPACING(S): _____ EXPLOSIVE TYPE(S): _____

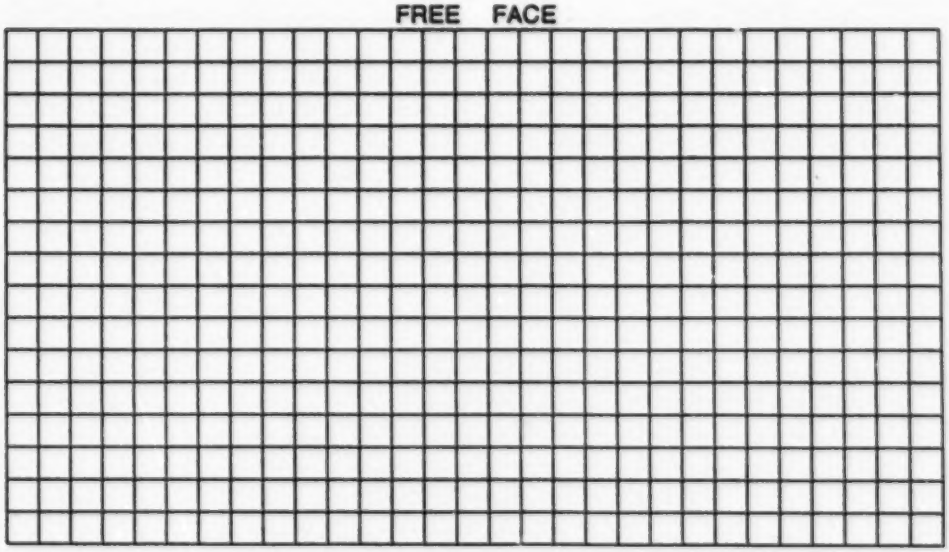
_____ _____

POWDER FACTOR: _____ CHARGE(S)/HOLE: _____

AVG. NO. OF HOLES: _____ INITIATION SYSTEM: _____

## SKETCH OF TYPICAL BLAST CONFIGURATION

FREE FACE



KEY

X Blasthole -5- Delay number (ms) --- Detonating cord

# PRODUCTION BLASTING SUMMARY
(PAGE 2 OF 2)

## ROCK MASS DESCRIPTION

ROCK TYPE(S): _____

_____

_____

DENSITY(S): _____

_____

_____

UNIAXIAL COMPRESSIVE STRENGTH(S): _____

_____

| STRUCTURE: | ORIENTATION | FREQUENCY |
|---|---|---|
| | _____ | _____ |
| | _____ | _____ |
| | _____ | _____ |
| | _____ | _____ |

## BLASTING EVALUATION

FRAGMENTATION: _____

_____

FREQUENCY OF SECONDARY BLASTING: ___/10

PRESENCE OF HARD TOES: _____

COMMENTS: _____

_____

_____

_____

_____

_____

COMPLETED BY: _____

POSITION: _____

DATE: _____

# CONTROLLED BLASTING SUMMARY

(PAGE 1 OF 2)

## DESIGN PARAMETERS

HOLE SIZE(S): _____ BENCH HEIGHT: _____

BURDEN(S): _____ SUBGRADE: _____

_____ STEMMING: _____

SPACING(S): _____ EXPLOSIVE TYPE(S): _____

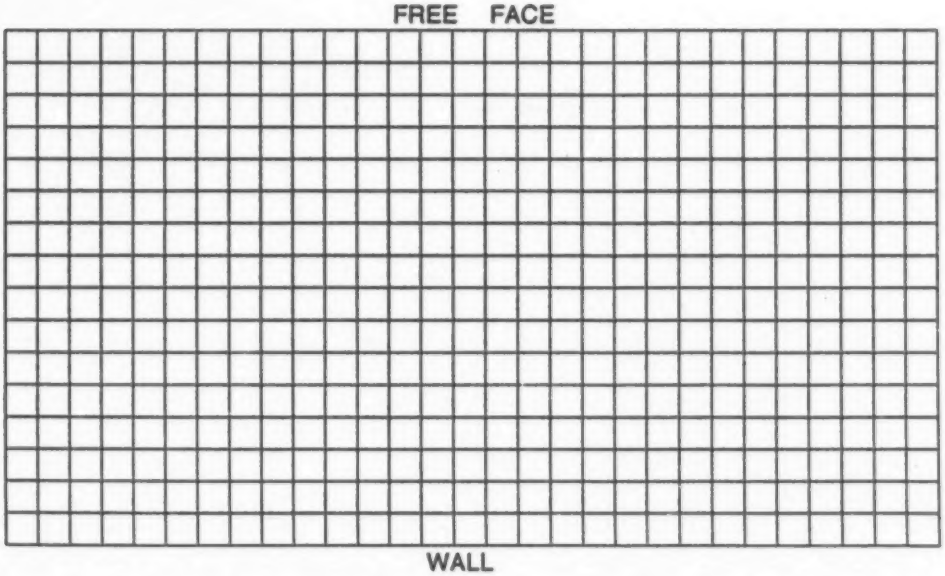
_____ _____

POWDER FACTOR: _____ CHARGE(S)/HOLE: _____

AVG. NO. OF HOLES: _____ INITIATION SYSTEM: _____

## SKETCH OF TYPICAL BLAST CONFIGURATION

FREE FACE



WALL

KEY

X Blasthole -5- Delay number (ms) ---- Detonating cord

# CONTROLLED BLASTING SUMMARY
(PAGE 2 OF 2)

## ROCK MASS DESCRIPTION

ROCK TYPE(S): _____

_____

_____

DENSITY(S): _____

_____

_____

UNIAXIAL COMPRESSIVE STRENGTH(S): _____

_____

| STRUCTURE: | ORIENTATION | FREQUENCY |
|---|---|---|
| | _____ | _____ |
| | _____ | _____ |
| | _____ | _____ |
| | _____ | _____ |

## BLASTING EVALUATION

ARBITRARY DAMAGE LEVEL:
(AS PER ATTACHED TABLE) _____

COMMENTS: _____

_____

_____

_____

_____

_____

**Table 1 – Levels of Blasting Damage Commonly Observed on Pit Walls**

| ARBITRARY DAMAGE LEVEL | JOINTS AND BLOCKS | DIP ANGLE AND APPEARANCE OF FACE | DIGGING CONDITION AT FACE |
|---|---|---|---|
| 1<br>Slight | Joints closed, infilling still welded. | >75<br>If used, semi–cylindrical sections of wall control blastholes seen. | Scars of shovel teeth seen in softer formations further digging not practical. |
| 2<br>Moderate | Weak joint infilling is broken, occasional blocks and joints slightly displaced. | >65<br>Face is smooth, some blast–hole sections seen. Minor cracks. | Some free digging possible, but teeth "chatter". |
| 3<br>Heavy | Some joints dislocated and displaced. | >65<br>Minor spalls from face. Radial cracking evident. | Free digging possible <1.5m with some effort. |
| 4<br>Severe | Face shattered, joints dis–located. Some blocks disoriented. | >55<br>Face irregular, some spalls, some backbreak cracks. | Free digging possible <3m. |
| 5<br>Extreme | Blocks dislocated and disoriented. Blast induced fines observed. | 55>37<br>Face highly irregular, heavy spalling from face. Large cracks behind crest. | Extensive free digging, >3m. |